PROPOSALS FOR
THE PROVING AND EXPLOITATION
OF THE BULLER GORGE URANIUM FIELD

A THESIS

PRESENTED TO THE UNIVERSITY OF NEW ZEALAND

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I. INTRODUCTION.

The discovery of uranium in New Zealand is perhaps the most significant mineral development since the discovery of gold about one hundred years ago.

In October 1955 two veteran prospectors, Messrs. Cassin and Jacobsen, discovered strong radioactivity in a road cutting near Hawks Crag, on the southern bank of the Buller River. The radioactivity was due to secondary uranium minerals occurring at the footwall of a porphyry dyke, in a fine grained breccia. The Government awarded Messrs. Cassin and Jacobsen a sum of £200 for their discovery. This grant was subsequently increased to £1,000 after the Atomic Energy Amendment Act was passed in 1957.

The discovery was followed by extensive prospecting in the area and early in 1956, prospectors employed by Nelson Lime and Marble Co. Ltd. found highly radioactive conglomerate boulders in the Buller River, in the vicinity of the original discovery. Chemical assays of samples from these boulders have yielded up to 2% U_3O_8.

The third, and so far the most significant discovery was the finding of a radioactive horizon in Uranium Creek, on the north side of the Buller River, at an elevation of approximately 1200 feet. This horizon (the "A" horizon) has been traced over a distance of over one mile. Subsequently, other radioactive outcrops, at higher elevations, have been located in the Hawks Crag Breccia. The work was done by prospectors employed by Mr. T. J. McKee (of Lime and Marble Ltd., Nelson) who holds Mineral Prospecting Warrants covering the area containing the uranium outcrops.

The Rio Tinto group displayed an interest in the uranium discoveries when in November 1956, geologists from Rio Australian Exploration Pty. Ltd., inspected the uranium outcrops and made an airborne scintillometer reconnaissance survey of the area. The results were generally negative, with an absence of both strong "background" and significant radioactive anomalies. Although they hold extensive applications for Mineral Prospecting Warrants over ground on the south side of the Buller River, no subsequent field interest has been shown by Rio Tinto.

Two geologists of the N.Z. Geological Survey have been in the area for the past two years. Aerial photographs were used extensively for mapping, for the compilation of a contour map, and for structural interpretations. Mines Department surveyors triangulated the area and mapped two of the radioactive horizons by stadia traverse.

The construction of an access road from the Tiriroa railway siding up the steep slope to Dee Point ridge (see Fig. 2) was financed by the Mines Department.

To date, the only consistent prospecting efforts have been made by the team employed by McKee, whose finances are limited.

Mining and geological officials from the United Kingdom Atomic Energy Authority have inspected the uraniferous horizons and the appointment to the area of one of their geologists in September 1958, probably means that proving of the deposit will be undertaken.

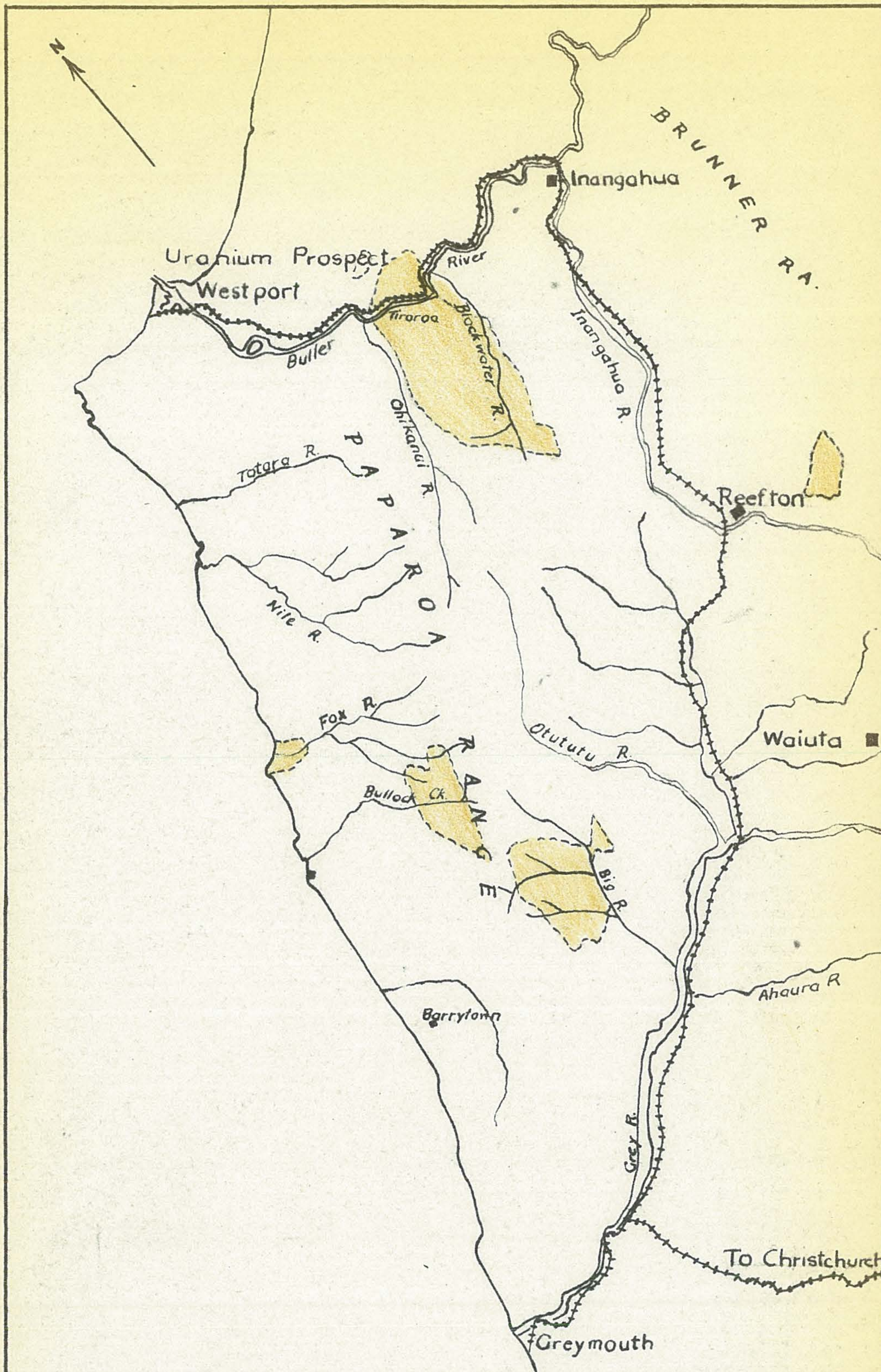

FIG 1 : LOCATION MAP SHOWING AREAS CONTAINING SOME URANIUM MINERALIZATION

Scale : 8 miles to 1 inch

II. GEOGRAPHY.

1. Location.

The regional location of the uranium deposit is shown on Fig. 1. Approximately 15-18 miles from Westport, one of the two towns of any size on the Westcoast, the uraniferous horizons present no serious problems of access. The road and railway line connecting Westport with the eastern seaboard of the South Island, pass through the Buller Gorge; the road on the south bank of the Buller River and the railway on the north.

With a population of approximately 5,000 inhabitants, Westport is the urban centre and port for the Buller District. The main industries in the district are coal mining, dairy and sheep farming, flax milling and the manufacture of cement (to be in operation in 1959). Port facilities are limited to ships of shallow draught - up to 5,000 tons displacement.

National Airways Corporation operate a daily air service between Westport and Wellington, but there is no direct air service to Christchurch, which route is served by twice daily railcars.

2. Climate, Topography and Vegetation.

The climate is temperate with a variation of only 10°F. to 20°F. temperature averages between summer and winter. There is a heavy rainfall of over 200 inches spread throughout the year.

The topography is extremely rugged and is dominated by the steep sided Buller Gorge, the sides of which rise precipitously to 2,500 feet. The drainage pattern consists of short streams of steep profile, sometimes eroding deep gashes in the hill side. Because of the steep slopes there is scant accumulation of the products of chemical weathering, and solid rock is found at the erosion surface in most places.

The area is covered by dense rain forest with the bush line at approximately 3,500 feet. The forest consists of southern beech, rimu, totara, rata and the usual undergrowth typical of the New Zealand rain forest. Since the area is a scenic reserve, the forest cannot be milled. However, Section 22, Clause (1) of the Mining Act, 1926, provides that timber may be felled for mining purposes on Crown lands which have been set apart as forest lands or reserves.

The heavy rainfall, rugged terrain, and dense bush make field work difficult, slow and expensive.

3. Power.

There are abundant resources of hydro-electric power in the South Island of New Zealand. Power reticulation is by means of a grid which connects all power stations in the island. If the grid system will not absorb the extra load caused by the requirements for mining and milling the uranium, then a coal fired generating plant could be constructed - utilizing the surplus of high grade bituminous fines available in the Buller Coalfield. Also the cost of the peak load, when connected to a grid, may be such that it is more economic to build and operate a coal fired power station at the mine.

Thus the power requirements of the operation will present no serious problems.

III. GEOLOGY.

1. General.

The geology of the Lower Buller Gorge has been described in various papers, the most notable being those by Wellman (1950) and Beck et al (1958). Only a brief outline is given here.

The stratigraphic sequence is shown in Table 1 and the geology of the mineralized area in Fig. 2.

TABLE 1.

Stratigraphic Sequence in the Lower Buller Gorge
(After Beck et al, 1958)

Name	Rock Type	Age
Tertiary	Non-marine at base	Eocene-Pliocene
U N C O N F O R M I T Y		
Porphyry Lamprophyre	Soda trachyte	Upper Cretaceous (?)
Hawks Crag Breccia	Conglomerate, breccia, sandstone, siltstone	Middle (?) Cretaceous
U N C O N F O R M I T Y		
Ohika	Shales, conglomerates, tuffs.	Jurassic
U N C O N F O R M I T Y		
Quartz porphyry		Jurassic (?)
Granite	Granite gneiss, diorite gneiss, pegmatite, hornfels, schist	Lower Paleozoic &/or Pre-Cambrian (?)
Greenland	Greywacke, phyllite, schist	Lower Paleozoic or Pre-Cambrian (?)

The uranium mineralization occurs in the Hawks Crag Breccia which has been mapped by the Geological Survey in some detail. Resting unconformably on the Ohika Beds, the Hawks Crag Breccia (H.C.B.) is thought to have been deposited in fault angle depressions (Beck, 1958); it is known to be separated from granite on the west by a major fault. The H.C.B. is of non-marine origin. The mostly unsorted greywacke, granite, and granite-gneiss fragments comprising it, range in size from pebbles up to 10 feet diameter boulders and are cemented together by a gritty matrix of much the same material. Well defined, but discontinuous bedding is confined to thin dark bands of fine grit and siltstone which are common in and near the mineralized horizons. Carbonaceous staining and plant remains are also found in these bands which are frequently highly radioactive.

After extensive field work in the Lower Buller Gorge area, Beck divided the H.C.B. into three facies, the division being made on the nature of the parent material. Table 2 shows the description of the facies and their areal extent can be seen in Fig. 2.

TABLE 2.

Facies in the Hawks Crag Breccia

Facies	Thickness (approx.)	Parent Rock	Type Area
Tiroroa B	2,500 ft.	Granite	South side, Buller R.
Dee Point	600 ft.	Greenland Greywacke	Hawks Crag, Dee Point.
Interfingering Zone	5-100 ft.	Granite & greywacke	Batty Creek
Tiroroa A	2,500 ft.	Granite	North side Buller R.
Blackwater	-	Granite & greywacke	Blackwater River

The uranium mineralization is mostly in the Tiroroa, or granite-derived facies: Tiroroa facies A contains 8 mineralized horizons. The horizon traced in the Radioactive Creek area (see Fig. 2) is in the zone of interfingering between the granite derived and greywacke derived breccia. Tiroroa facies bed B, outcropping in the south side of the Buller River over a large area, is a possible host for extensive uranium mineralization. Although little prospecting work has been done south of the Buller River, bedded uranium deposits have been found in a tributary of the Ohika-iti River (op. cit.).

Besides the bedded deposits of the Tiroroa facies there is some vein-type mineralization in the Lower Buller Gorge area. The Batty Creek "lode" (which is the original discovery of Cassin and Jacobsen) and radioactive quartz-pyrite veins in Mispickel Creek and Quartz Creek are examples of this.

Erosion remnants of the H.C.B. occur elsewhere on the northern part of the West Coast. The location of these outcrops is shown in Fig. 1. Uraniferous horizons have been discovered in the Bullock Creek area, in a host rock which closely resembles the Tiroroa facies (op. cit.). Slight radioactivity has been detected at the mouth of the Fox River and in the vicinity of the Waitahu River. Highly radioactive boulders have been discovered by prospectors in Big River.

Thus the areas of H.C.B. indicated on Fig. 1, are possible potential sources of uranium ore. But the area north of the Buller Gorge, held by Buller Uranium Ltd., contains the best known surface indications of uranium mineralization. Accordingly, it is in this area that a systematic sampling campaign is advocated.

2. Geological Factors Effecting the Mining and Proving the Deposit.

Eight mineralized horizons have been identified in the Uranium Creek-Bullen Creek area, although the "A" and "C" horizons are the only ones whose continuity appear to have been proved so far. The radioactive horizon in the Radioactive Creek area is probably higher by several hundred feet, than the top horizon in the Uranium Creek - Bullen Creek area (the "H" horizon, see Fig. 2).

Reed and Claridge (1957) identified the main primary uranium mineral as coffinite which is a hydroxyl - substituted uranous silicate. The coffinite is of fine grain and occurs interstitially in an intimate mixture with pyrite and carbonate (op. cit.). The age of the coffinite has been established as within the range 100 million to 150 million years (by kind permission of Rio Australian Exploration Pty. Ltd.). This places the coffinite age-range in the Cretaceous period which is the period when the Hawks Crag Breccia was deposited.

Radiometric prospecting of the outcrops showed extremely wide variation in grade even within six inches along the bedding. The variation in surface radioactivity may be partly due to inequilibrium through natural leaching of the outcrops.

The ore-grade mineralization, occurring at different elevations appears to be uneven in distribution; possibly corresponding to paleo-stream channels, and the values either following a dendritic pattern, as is the case of the gold-uranium bearing conglomerates of the Witwatersrand; or more likely in a series of discontinuous lenses of varying thickness lying at different elevations, similar to the distribution of the coffinite horizons at Ambrosia Lake, New Mexico (Young et al, 1956).

A study of the geological map, Fig. 2, reveals variations in the strike and dip. Rather than interpreting these as fold structures, Beck (1958) accounts for them by a general tapering of the beds (see Fig. 3), and the topography of the basal beds at the time of deposition. The angle of dip of the ore horizons varies from 5 to 35 deg. in a general south-west direction.

The average mining width is expected to be little greater than the minimum mining width of four feet. Consequently, the mining methods considered at this stage must be applicable to thin deposits.

The ore may, therefore, vary in grade, lateral extent, thickness and dip over relatively short distances. The mining method must be sufficiently flexible to be adapted to these changes.

The lowest horizon, the "A" horizon, is the thickest and most extensive so far found. The positions of the outcropping horizons are shown in Fig. 4, which was drawn up by the Mines Department from a stadia survey. The section along the line AB in the plan is shown in Fig. 5, in which the mineralized horizons have been idealized in that they are shown to be continuous and at constant dip. In fact this is unlikely to be correct and the significance of the section lies in the relative vertical positions of the horizons which is most important in the design of the mine layout. From the section it can be seen that the vertical distance between the "A" horizon and the "H" horizon is approximately 600 feet. To mine all horizons a large amount of development work will be required. It is obvious that there must be ore, of sufficient tonnage and grade, in each horizon to cover the cost of mining it, and the development work required to open it up for mining.

Faults are common in the area. All the faults dip steeply, some are large with crush zones up to 20 feet wide, but most appear to be of relatively small displacement. From a study of the fault pattern as determined from aerial photographs and subsequent field work, it is concluded that the two main strike directions of the faults are 030 and 120 deg. true (Mines Dept.) A more detailed study of the surface geology will not yield much more information in this respect because of the lack of marker horizons. Any underground workings in large crush zones and closely faulted areas, will necessitate close timbering. Core recovery, when core-drilling through faulted country, will be low and difficulty may be experienced in drilling because of water losses in the shattered rock. It is important that both exploration drilling and underground development in fault-shattered areas be avoided.

Although the rock fragments have become sub-rounded in shape by mechanical weathering during transportation from the source, little or no chemical weathering appears to have taken place at the source. Even the felspar fragments examined showed no signs of "rotting" around the edges. This fact is advantageous from both milling and mining aspects. Weathered rock (especially the clay minerals) has an adverse effect on crushing and grinding operations. With no products of chemical weathering and the rock fragments well cemented together there should be little danger from rocks spalling off the backs of unlagged underground openings.

Since there are few well-defined continuous partings, or bedding planes the rock can be expected to behave more like a homogeneous igneous rock rather than a sedimentary type of rock, having more or less equal strength in all directions. This statement requires some qualifications as the strength of the rock is dependant on the nature of the matrix which varies from coarse grit to fine siltstone. The rock is weakest where the matrix is coarsest. Generally, the ground appears to be strong and rather than caving easily, shearing in the weaker matrix is the expected mode of failure when excess pressure is applied. That is, the rock will tend to "block off", rather than bend and cave to any particular pattern.

Rock hardness is obviously variable because of the different constituents present in the breccia. The hardness can be expected to vary from quartz (Moh's hardness No. 7) through granite, to the softer matrix, with a Moh's hardness of 3-4, Therefore, bit wear and penetration speeds in drilling will be variable. In the estimation of drilling performances and costs, an average hardness corresponding to that of granite (Moh's hardness of 5-6) is used.

IV. LEGISLATION AND TITLES.

1. General.

Legislation governing the prospecting, mining and marketing of radioactive minerals is mainly contained in the Atomic Energy Act 1945 and the Atomic Energy Amendment Act 1957.

The Atomic Energy Act of 1945 was of a general nature, being enacted at a time when the world was in a state of panic over the power of the atomic bombs dropped on Hiroshima and Nagasaki. At that time, too, the possibility of finding uranium in New Zealand seemed remote.

By exercising its right of eminent domain in the 1945 Act, the Crown became the owner of all uranium in its natural condition in the ground, regardless of whether the land was held in fee-simple or not. This was desirable for both technological and strategic reasons and was in accordance with legislation passed in most countries of the world at about that time.

Mining and prospecting were inadequately dealt with in the 1945 Act and little incentive was given either to prospectors or organised geological exploration. The only goal that uranium prospectors could look forward to was a grant by the Minister of Mines (the amount of which was at his discretion) for the first uranium discovery. [section 4, A.E. Act 1945]

The Crown was given the power to control the mining and concentrating of radioactive minerals, and the extracted uranium became the property of the Crown, to be disposed of as the Minister of Mines directed. The price paid by the Government for uranium mined in New Zealand was "to be fixed by the

Minister of Mines having regard to the costs of production and to such other factors as may, in his opinion, be relevant."

Following the initial discovery of uranium in the Buller Gorge in 1956, there was intense activity in uranium prospecting and geological exploration in that area, and the shortcomings of the 1945 Atomic Energy Act became increasingly apparent. In October 1957 Parliament passed an Act to amend the 1945 Act. The Amendment closed most of the gaps in the existing legislation, granting more favourable terms to prospectors and to uranium exploration generally. It also limited the power of the Minister of Mines as defined in the 1945 Act.

The Mining Act 1926, applies to uranium mining, except where otherwise stated in section 6 of the Atomic Energy Amendment Act 1957.

2. Exploration and Prospecting.

In the 1957 Amendment, sub-section 2, section 4 of the 1945 Act was repealed, and more realistic awards were provided to encourage uranium prospecting.

1. A grant of up to £200 for the discovery of a deposit of "prescribed substance"* which is not of immediate commercial value, but is of sufficient geological interest to justify further prospecting.

2. A grant of up to £1000 for the discovery of a deposit of "prescribed substance" of potential value.

3. If the deposit contains 25 tons or more of uranium oxide (U_3O_8), then in addition to the £1000, the discoverer of the deposit receives £400 for each 5 tons of uranium oxide, that the deposit is estimated to produce.

These grants are exempt from income tax and the social security charge.

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* "Prescribed substance" is defined as any substance which may be used for the production of atomic energy or research into matters connected therewith, e.g. uranium, thorium, plutonium, neptunium.

In Section 4 of the 1957 Amendment Act, provision is made for the financial assistance of persons prospecting for "prescribed substances". Any Government grant may be in the form of cash payment, loan or subsidy.

The holder of a miner's right or mineral prospecting warrant may prospect for "prescribed substances" The miner's right is merely a personal qualification authorizing the holder to prospect on Crown land. A mineral prospecting warrant, for "prescribed substances" may be obtained on application, from the Warden's Court. This gives the holder the exclusive right to prospect for the "prescribed substances" only, on the area granted him which is not to exceed 10,000 acres, for a period of 5 years. The rent payable under a mineral prospecting warrant is 1d. an acre per annum for the first two years, 2d. for the third year, 3d. for the fourth year and 6d. for the fifth year.

Section 18 of the Mining Regulations as amended by clause 2 of Amendment No. 13 1954, provides that:

> "within three months after the issue of a mineral prospecting warrant the holder shall commence a geological, geophysical, or other investigation of the mineral deposits of the area comprised in the warrant, and shall continue the investigation with reasonable diligence and skill while the warrant is in force."

This is to ensure that mineral prospecting warrants are not being held in the hope that they become more valuable, if an economic mineral deposit is found on an adjoining property. The 1954 Amendment takes a more reasonable attitude towards modern geological exploration techniques by replacing a regulation which imposed an obligation to employ workmen when a mineral prospecting warrant was granted.

The holder of a mineral prospecting warrant is required every six months to give a written report to the local Inspector of Mines, on the amount expended, and the nature of the exploration work done, during the previous six months.

3. Mining.

Sub-section 5, section 6 of the 1957 Amendment Act provides that a mineral license, granted under section 106 of the Mining Act 1926, is necessary before uranium operations can commence.

The holder of a miner's right may apply for a mineral license, whether or not he is in possession of a mineral prospecting warrant. The holder of a mineral prospecting warrant has priority in obtaining mineral licenses on the area he holds. Although there appears to be no limit to the number of mineral licenses that can be held, each license is limited to an area not exceeding 320 acres for each one thousand acres (or part thereof) comprised in the mineral prospecting warrant.

A mineral license for "prescribed substances" under subsection 6, section 6, Atomic Energy Amendment Act 1957 authorizes "the licensee to occupy any land to which the license related whether or not it is Crown land and whether or not it is situated in a Mining District." The written consent of the landowner and lessee must be given, however, before the license is granted.

4. Marketing.

The marketing of uranium is placed on a more realistic basis by the repeal of section 6 of the Atomic Energy Act 1945, under which all extracted uranium must be delivered to the Crown, at a price determined by the Minister of Mines. Section 7 of the 1957 Amendment Act allows the uranium producer to sell his product to buyers who are approved by the Minister of Mines, and "subject to such conditions as he shall impose." The Crown, however, retains the right to acquire the uranium mineral or concentrate from the producer if it should desire to do so at any time. The price paid by the Government in such a case, is the price which the owner of the uranium might reasonably have been expected to obtain by selling his product immediately before the date he receives notice that the Crown is acquiring his uranium.

The uranium producer is, therefore, allowed some freedom of action in his marketing, and in the event of his uranium being acquired by the Crown, he is assured of a fair market price.

5. Titles.

The following current Mineral Prospecting Warrants held or applied for in areas of Hawks Crag Breccia are listed in the N.Z. Mines Statement, 1957:

Number.	Name.	Location.	Area.
13485	Cassin & Jacobsen	Ohika Survey District	54 acres
13486	Lime and Marble Ltd.	Ohika Survey District	1360
13492	" " " "	" " "	960
13493	" " " "	" " "	565
13514	" " " "	" " "	400
13541	" " " "	" " "	1632
13678	" " " "	Kawatiri Survey District & Ohika " "	1960
13497	Munden and others	Ohika Survey District	800
13552	J.S. Smith	" " "	200
13630	L.D. McAlister	Ohika Survey District & Kawatiri " "	1000
13735	J. W. Fair	Ohika " "	1000
Application No. 113/56	G.J. Williams	South side Buller Gorge	
Application Nos. 123/56, 124/56, 125/56 & 4/57	R. Searle	South side Buller Gorge	
13736	McAlister & Dowd	Fox River Mouth Brighton Survey District	1000
13737	McEnaney & others	Headwaters Fox River Brighton Survey District	1240
No. 29/1957	Lime & Marble Ltd.	Waiwhero " "	5195
No. 30/1957	" " " "	Waiwhero " "	7329
Application No. 45/1957	Lime & Marble Ltd.	Brighton, Waiwhero, Mawheraiti Survey District	9900

V. TAXATION.

1. Royalty.

The royalty may be regarded as a payment made by the mining company in return for being allowed to extract irreplaceable minerals from the ground. The royalty constitutes an expense of production and therefore must be taken into account in the early stages of development, when the cut-off grade of the ore is being determined.

Royalties on uranium are payable under the provisions of the Mining Act 1926. Sub-clause (e), section 106, of the Mining Act 1926, as amended by Clause 2 of the Mining Amendment Act 1948, provided:

> "The licensee shall also pay in respect of all the specified metals and minerals raised pursuant to the license, such royalty as is specified fixed by reference to their weight or quantity."

The amount of royalty payable is not stated. The legislation implies that the royalty is calculated at a flat rate on production, for example, 2/- per ton of ore mined. A royalty of this type, in effect, raises the cut-off grade of ore and so reduces the ore reserves. This is detrimental to both the mining company and the nation. For this reason it would be to New Zealand's advantage to abolish royalties on uranium from the suspected low grade Buller deposit.

2. Income Taxation.

These days income taxation on mining profits is so considerable as to be a factor of major importance in planning mining investment. The cost of taxation to a mining company is dependant only on the existing legislation. The degree of enlightenment shown in the legislation, often decides whether a low grade mineral deposit is an economic proposition or not. There are many cases of this throughout the world. For example, the existence of low grade base metal deposits in the British

Isles has been known for many years, yet those in the Republic of Ireland are worked while those in England are not. The only reason for this is the difference in the taxation systems of the two countries.

Mining investment differs from other forms of business enterprise (with the exception of the petroleum industry) in that the prime asset, i.e. the mineral deposit, is gradually consumed throughout the life of the mine. In the taxation legislation of most countries some provision is made, therefore, for the return of the capital invested, to the shareholder. This is effected by a deduction (usually a percentage of the nett profit) from the nett profit in the calculation of the taxable income, to allow for the depletion of the mineral reserves. In addition expenses incurred in exploration, prospecting, proving and developing a mineral deposit are deducted from the nett profit, not necessarily in the year in which they are incurred but over a specified period during which the mine is at full production. In the initial stages of the development of the deposit these expenditures greatly exceed any revenue from mining.

The mining countries of the British Commonwealth have developed taxation systems which provide incentives where they are most needed to keep their respective mining industries flourishing. In Canada, the inherent financial risks of mining propositions are fully recognised through taxation incentives. The initial three year tax free period and comprehensive depletion and depreciation allowances render Canada's mining taxation legislation the most enlightened in the world.

It is worthy of note that in Australia, uranium mining companies in which at least three quarters of the voting power is controlled by Australian residents, are exempt from taxation. This is one example of a taxation incentive designed to attract capital from local sources rather than overseas.

In New Zealand, Section 152 of the Land and Income Tax Act 1954 provides a depletion deduction for the mining of gold, mercury, scheelite, molybdenite, lead, zinc, antimony, tin and manganese. Companies deriving their income from the mining of these metals are taxed on an income assessed as one half the dividends paid to the shareholders during the year, if the total dividends paid since the company's inception do not exceed twice the amount of capital paid up in cash. Otherwise the company is taxed on an income equal to the dividends paid during the year.

Clause 12 of the Land and Income Tax Amendment Act (No. 2) 1957, provides that section 152 shall apply to uranium.

This deduction for the depletion of reserves compares favourably with that in other countries. It enables an investor to recover his initial capital, provided the mine life and profits are sufficient. If half the dividend is regarded as the interest on the investment, it is rightly subject to tax, the other half being regarded as the return of capital is tax free. This continues until the investor receives a sum equal to his original investment.

Since the taxable income at the most is limited to the dividends paid to the shareholders, all expenditure, either direct or computed costs, incurred by a uranium mining company may be deducted from gross profits. This would allow a company engaged in mining and treating the Buller Gorge ores, considerable financial latitude in the depreciation of assets, and in distributing such large expenditures as the cost of the building, treatment plant, ventilation system and the cost of exploring, proving and developing the deposit, over the profitable production years.

The only limit to the use of profits for development, equipment, social services, lies in the reasonable requirements of the shareholders for the payment of interest on their investment: when profits are distributed to shareholders, the depletion allowance is applied and the remainder is subject to income taxation.

Thus, the company can build up its financial reserves and material assets over the first few years of profitable operations without any pressure from taxation demands.

It appears then, that a company mining uranium would operate under extremely favourable income taxation conditions in New Zealand.

<u>Note</u>: Since mining companies are taxed only on distributed profits (dividends) the excess profits retention tax which applies to New Zealand companies as from 1958, is not applicable to mining companies (according to N.Z. Inland Revenue Department, Dunedin Branch).

VI. SURVEY OF THE URANIUM MARKET.

1. The Development of the Industry.

It was not until 1939 that any economic or strategic significance was attached to uranium. Prior to that time, uranium was mined as a by-product of radium, its use being as a pigment in industry. The small amount of uranium required was obtained from the known deposits of Joachimstahl (Czechoslovakia), Shinkolobwe (Belgian Congo), and the Great Bear Lake (Canada).

In 1939 it was found that when the nucleus of the isotope U^{235} is made to absorb an additional neutron it undergoes fission. That is, each nucleus is split into two parts which are thrown apart with the consequent generation of heat from the energy of motion. During the fission process two or three neutrons are absorbed in neighbouring U^{235} nuclei and produce further fissions. This process is repeated to form the chain reaction.

The first application of the fission chain reaction was in the Atomic bombs dropped on Japan to end World War II in 1945. The terrifying success of the atomic bombs made the world fully conscious of the strategic importance of uranium.

It became vital to the military purposes of the Western Powers (namely the United States and United Kingdom) that they have large resources of uranium. Perhaps more important was the need to gain control of all uranium production.

The British-American Atomic Energy organization extended its exploration activities to all potential sources of uranium. The adaption of the Geiger-Muller Counter as a prospecting device, and the widespread occurrence of uranium mineralization also gave impetus to the rapid discovery of uranium deposits.

The economics of uranium mining had little significance.

A ready market was assured by the Atomic Energy Authorities of the United Kingdom and United States, who either jointly or separately entered into long term, fixed price contracts, with all the free world producers. These contracts were negotiated individually with each producer, on the basis of estimated costs of production and an adequate profit margin. Provision was also made in the contracts for the complete amortization of the capital required for the mill and mine development, during the tenure of the contract.

With loan finance supplied by the Atomic Energy Authorities the potential uranium producer did not have to raise large aggregates of capital. Further, the guaranteed market eliminated price fluctuations which greatly add to the risks involved in the mining of most metals. This was all the impetus potential producers required and there was an unprecedented expansion of the uranium mining industry.

In the early stages, it was thought that the mining industry could not keep pace with the apparently insatiable demand for uranium, created by the ever present threat of nuclear warfare. By 1957 however, the contracted supply of uranium was more than sufficient to meet military requirements, and no new contracts have been entered into by either the United States or the United Kingdom since that time. Thus the uranium mining industry has undergone spectacular and tremendous expansion. Since the end of World War II it has grown from an industry of modest proportion to become the second largest metal mining industry; in monetary value it is second only to gold production in the Free World, with an estimated production of 40,000 tons of uranium oxide in 1959.

Uranium producers must now turn to a less urgent and vital, but nevertheless steadily increasing market for their product - the nuclear power field. The first atomic power station was brought into operation in October 1956 at Calder Hill in Great Britain, and there are more projected throughout the world.

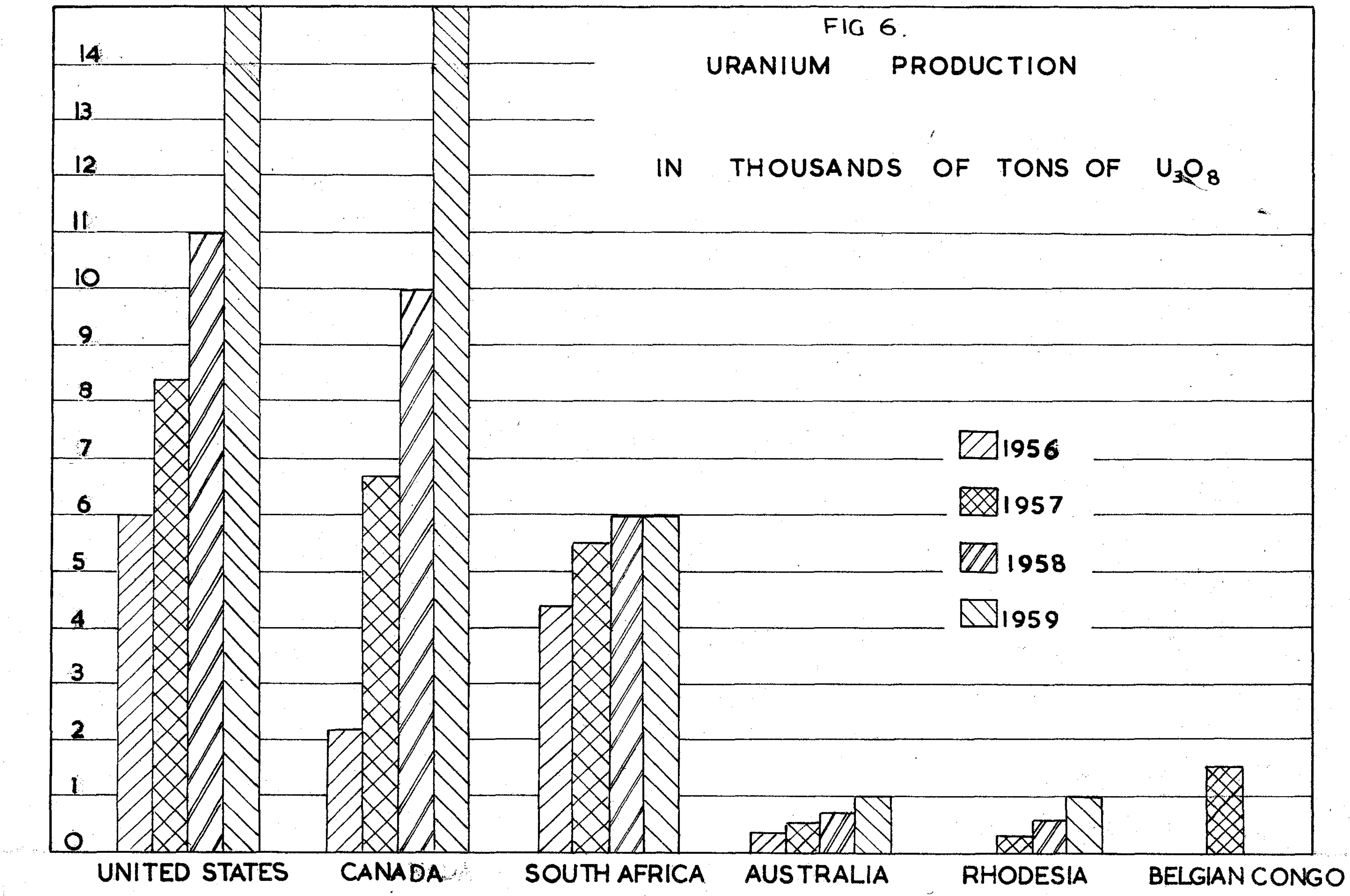


FIG 6.
URANIUM PRODUCTION
IN THOUSANDS OF TONS OF U_3O_8
1956
1957
1958
1959
0
1
2
3
4
5
6
7
8
9
10
11
12
13
14
UNITED STATES
CANADA
SOUTH AFRICA
AUSTRALIA
RHODESIA
BELGIAN CONGO

2. Supply.

Fig. 6 shows the production, in tons of U_3O_8, of the major uranium producing countries. The estimated production for 1959 constitutes the maximum production provided for by current contracts. It is reasonable to assume that production will remain about this level of 40,000 tons of oxide per year - at least until the contracts expire in the middle 1960's.

TABLE 3.

Published Uranium Reserves of the Free World as in 1957.

	Millions of tons of ore	$\%U_3O_8$	Tons U_3O_8
United States	75	0·27	172,500
Canada	320	0·12	384,000
South Africa	1,100	0·03	330,000
Australia	0·230	0·3	700
France	0·1	0·2	200
Italy	3	0·2	6,000
			893,400

Table 3, showing the published reserves of uranium requires some qualification.

The figures quoted in the table were abstracted from various mining journals and it has been impossible to ascertain how these reserves were established. The extent of the knowledge upon which the estimates were made is clouded further by the general confusion in the nomenclature and classifioation of ore reserves. For instance, the 75,000,000 tons quoted as U.S. ore reserves are developed reserves (Youngberg 1958). The Canadian estimate of 320,000,000 tons is probably measured and indicated ore since the inferred ore (based largely on geological interpretatiom of structure) of the Blind River field alone is about 500,000,000 tons. In South Africa blocked out reserves total

64,000,000 tons and the ore reserves in the table includes inferred ore. The Australian ore reserves quoted in the table are proved reserves from Rum Jungle only.

It can be seen that some countries do not appear to have sufficient reserves to maintain their present production rates until 1975. For example, the United States has sufficient developed reserves to last only 10 years at present milling capacity. It is not usual for an orebody to be completely delineated before mining commences. Since the mines have been producing for five years at the most, it is reasonable to assume that the reserves may be increased as underground development is carried out in the course of normal mining operations. An example of this is the extensive ore discoveries reported from the Radium Hill and Rum Jungle mines in 1957.

One general conclusion that can be drawn is that the total Free World reserves of over 800,000 tons of U_3O_8 are sufficient at present production rates until after 1975. It is interesting to note the strategic location of the major uranium fields. With the possible exception of South Africa, all are situated in countries where there is little likelihood of political unrest.

There are other potential producers of uranium which will supply the fuel for domestic nuclear power programmes. The following potential producers were listed in the Mining Journal Annual Review Issue of 1958. In France, production is expected to increase to a maximum of 1,000 tons of U_3O_8 per year by 1961. An unrevealed amount of uranium is being mined in Portugal. The large reserves which have been established in Italy are expected to ultimately yield 350 tons of U_3O_8 per year. Two mills are being constructed in Brazil where 0·2% U_3O_8 ores have been located. In Labrador a high grade deposit (about 0·75%) is being developed.

The discovery of many new deposits has been reported in the past two years; some of these are in Greenland, Fiji, Japan, Switzerland, Madagascar, Morocco, Peru, and Chile. This emphasises the widespread occurrence of uranium mineralization.

Experience in recent years has proved that the mining industry can develope ores for mining at a much faster rate than the uranium can be consumed. A better understanding of uranium geology and improved technology has made this possible. Also, because of the shallow depths of many of the large bedded deposits, it has been possible to obtain a high rate of ore production quickly.

There is little doubt that the uranium reserves of the Free World could be greatly increased if the demand existed. From this point of view alone the favourable circumstances under which the producers operate at present are unlikely to be repeated.

3. Military Demand.

A survey of the uranium market is made difficult by the uncertainty of the demand for military purposes.

The uranium industry owes its rapid growth entirely to the military demand which exists because of the state of international politics. The United States has contracted for about 80% of the Free World production until the middle 1960's (Libby, 1957). All of this is absorbed in the American diffusion plants which separate the fissile U^{235} from the raw uranium. These plants are working largely for military purposes, the U^{235} and plutonium ultimately going to the strategic weapons stockpile. Thus the military demand is dominated by the United States. The recent restrictions imposed by the United States Atomic Energy Commission (USAEC) on domestic production (referred to in more detail later), together with the fact that the USAEC has not entered into any new foreign uranium contracts for the past few years, suggests that the military demand is satisfied.

If the military demand was suddenly increased under the threat of imminent nuclear warfare, any increase of production would be required quickly. The increased demand could most readily be met by expansion of operations on the part of existing producers - rather than developing a new field such as the Buller Gorge.

Regarding general nuclear disarmament and its effect on the uranium industry, Sir Edwin Plowden, Chairman of the United Kingdom Atomic Energy Authority, stated in December 1957:

> "..... If on the other hand there is general nuclear disarmament, the demand for uranium will obviously fall for a time. But it would be pessimistic to deduce that the longer term effects would be bad for the uranium mining industry ..... Nuclear weapon programmes exact a high price in money and in the absorption of technical resources from the countries which undertake them. If there were nuclear disarmament, these resources would be available for civil purposes. Doubtless a direct acceleration of the civil uses of atomic energy would result....."

With no obvious diplomatic moves being made towards nuclear disarmament the most reasonable assumption to make is that the military demand will continue at about the same level at which it is now.

Although the United States and the United Kingdom have obviously planned to absorb their contracted amounts of uranium, it is not likely that stockpiling will continue indefinitely. An international agency has already been set up to lend a substantial store of fissile materials to those countries which guarantee to use it for peaceful purposes. Supplied mainly by the United States but partly by the United Kingdom and U.S.S.R., this material will presumably come from the strategic weapons stockpiles of fissile materials. This is an additional complication in that part of the military requirements may supply a civilian demand.

By analogy with base metals, strategic stockpiles can act as a buffer between supply and demand. Providing uranium is on the open market it can be sold to consumers when supplies are scarce. Conversely the uranium can be bought for the stockpile when the market is saturated.

4. The Power Resources of the World.

According to Carman (1957) the rate of increase of world energy output during the last decade was 6% per annum. This means that by say, 1980 (if current rates of growth are maintained) the world energy requirements will be about four times the present output. To supply this future power demand there are the fossil fuels (coal, petroleum and natural gas), hydro-electric power and nuclear energy.

Coal has been the mainstay for the power requirements of the great industrial countries of the world since the industrial revolution. It is generally agreed that in these countries at least, workable reserves of coal are not sufficient to meet current requirements (except in the United States where reserves are over 1000 million tons). Admittedly there are large reserves of coal in the more undeveloped countries. But in general, transport costs preclude their exploitation. The increasing number of chemical products made from coal may also limit its use as a fuel.

With regards to petroleum the following is quoted from Carman (1957).

"..... Proved reserves of petroleum total about 30 years supply at current rates of production. There is reason to believe that the ultimate reserve may be about 5 times this amount (the reliability of this statement is questionable). ..... Between now and the year 2000 it is estimated that the energy requirements of transportation may increase tenfold. In actual fact there is every possibility that one day soon nations will take action to outlaw petroleum and natural gas as fuels for generating electricity. Such a rapidly dwindling resource quite properly should be reserved for the petro-chemical industries....."

If any reliance can be placed on the estimates of Carman, then the future of petroleum as a fuel associated with the generation of electricity, is likely to be short.

Perhaps more important for the immediate future is the strategic position of most of the world's petroleum reserve. The Suez Crisis of 1956, the current unrest in the Middle East and the vulnerability of the Middle Eastern oil producing countries to communist domination, must act as a deterrent to the construction of oil fueled power stations in the Free World.

The third source of power is hydro-electricity. It is considered capable of the greatest expansion of the power sources considered so far. The greater proportion of the world's water power potential is south of latitude 20 deg. N. (N.Z.B.S. - Atoms for Peace Programme Aug. 1958), while the great industrial countries are north of this latitude. Africa, South-east Asia and South America have 73% of the world's water power potential and they have exploited only 3% (Carman). The main reason for the non-development of these water power resources is the high capital cost of hydro-electric schemes. For example Worley (1954) estimated the capital cost of Roxburgh, N.Z., which has a generating capacity of 320,000 kilowatts as £77 per K.W. Foreign aid programmes (such as the ill-fated Egyptian Aswan Dam project) and large foreign industrial projects (the possibility of aluminium smelting in New Guinea) are ways in which this potential can be developed.

The more developed countries have harnessed most of their available water power. The only untapped water resources are generally of such location and character that it is cheaper to operate thermal power stations.

5. Civil Demand.

The major demand for uranium for peaceful purposes will be for use in nuclear reactors associated with the generation of electricity. In order to understand the uranium demand created by nuclear power programmes, some general principles of operation of nuclear reactors are given below.

The isotope U^{235} is a fissile material (i.e. it undergoes nuclear fission), but it forms only one part in 140 of natural uranium which is mostly U^{238}. U^{238} is termed a fertile material since although it contains a power potential about equal to that of U^{235} it can only be used indirectly. To become a fissile material the U^{238} is converted into plutonium-210 by the absorption of neutrons. Goodlet (1958) describes the reactor as performing two functions:

1. The heat produced in the nuclear fission process is used to produce steam for the generation of electricity.

2. Only 40% of the neutrons released in fission are required to maintain the chain reaction at a steady rate. The surplus neutrons (comprising 60% of the total) can be used to produce new nuclear fuel by the conversion of the fertile substances, U^{238} and

Th^{232}, into fissile material. Since more neutrons are available for conversion than are needed to produce fission, in ideal conditions more reactor fuel can be produced than is consumed. Reactors which can do this are termed breeders.

According to Thomson (1957) the net effect of running a reactor which consumes natural uranium, is to burn some of the U^{235} and to turn some of the U^{238} into plutonium. Some of this plutonium is burnt but most is left behind. This process cannot go on indefinitely. Other products are formed and the uranium pile becomes poisoned, also the fuel rods undergo mechanical damage and must be replaced. Thomson also indicated the small amount of the original material actually consumed in these reactors. He considered a reactor in which the uranium was used until the U^{235} had been reduced by 33% and then rejected, after separating out the plutonium (which is less than the amount of U^{235} consumed). Even if the plutonium was burned later the reactor would be consuming the equivalent of less than $\frac{2}{3}$ of the original U^{235}. That is, less than 0·5% of the original uranium fuel element. In practice, however, the rods may be removed after only 0·1% of the uranium has been used. Research is currently directed towards increasing the percentage burnt or "burn-up" in this first stage. The rejected uranium may be used again if enriched by U^{235} or plutonium.

The low "burn-up" achieved by those reactors using natural uranium, leads to a relatively inefficient use of uranium fuel. The economics of fuel enriched by the addition of U^{235} or plutonium is being studied at present. Enriched fuel implies the use of diffusion plants to separate out the U^{235}. Separation of the U^{235} is an expensive process and the United States is at present the only country with such plants. Although natural uranium is used to produce the U^{235}, reactors using enriched fuel will consume less uranium than those using natural uranium, for a given amount of electricity. It is now well established that in certain types of reactors using highly enriched fuel it is possible to breed fissile material. The feasibility and economics of breeder reactors, using thorium (which is apparently particularly suitable as a fuel in breeder reactors) are being investigated in the United States and Great Britain. They will require large initial charges but their running requirements are anticipated to be extremely small. Thomson has estimated that the consumption of 30% of the original material will be possible in breeder reactors.

From the foregoing it can be seen that the type and amount of fuel charges required in nuclear reactors differs greatly from that in conventional coal or oil fired power stations. A reactor requires a large initial charge and thereafter only a small proportion of the fuel charge is removed each year and has to be replaced. <u>The rate of commissioning of power stations will therefore be the important factor in uranium consumption</u> - rather than the actual installed generating capacity at any particular time. This is especially true of reactors using enriched fuel and breeder reactors. With "burn-up" varying from 0·1% to an estimated 30% in breeder reactors, any attempt to estimate the quantity of uranium likely to be used for replacement purposes, can only be a guess.

The United Kingdom is the only country with an operating nuclear power station and a clearly formulated nuclear power programme. Accordingly, Calder Hall and the projected power stations in the United Kingdom are discussed in some detail as to fuel consumption and operating costs. The United Kingdom power stations will all use natural uranium as a fuel. The Calder Hall power station was not designed primarily for power production, but to produce plutonium for military purposes, with 70MW of electricity as a by-product. The projected British nuclear power programme announced in March, 1957, is for 5,000-6,000 MW of nuclear capacity to be installed by 1965.

This means that in 1965 15% of the generating capacity of the United Kingdom will be nuclear; but as the nuclear stations will be base load stations they will supply about 25% of the electricity actually used.

Goodlet (1958) has calculated the uranium metal requirements for the United Kingdom nuclear power programme. He has assumed a 28% overall thermal efficiency, and has based his calculations on the fact that it is possible to achieve a heat rate of 2 MW per tonne (≡ 1,000 kilograms) using gas-cooled reactors.

At 28% efficiency 6,000 MW of electricity = 21,500 MW of heat energy

At 2 MW per tonne the <u>initial uranium charge</u> = 11,000 tonnes
= 10,800 tons.

Considering full output for 75% of the time, i.e. 274 days per year
Heat energy produced = 21,500 x 274
= 6,000,000 MW days.

Since the amount of fission heat that can be extracted from one tonne of uranium, before it has to be replaced in the reactor, is expected to be about 3,000 MW days (this corresponds to a "burn-up" of 0·3%)

Annual consumption = 2,000 tonnes
= 2,000 tons.

∴ Initial charge per MW of generating capacity = 1·8 tons.
Annual replacement" " " " " = 0·33 tons.

<u>N.B.</u> The weights refer to the finished fuel elements and do not allow for manufacturing waste etc.

The above figures may be too large, since it should be possible to increase the "burn-up" rate in the future. One tonne of the spent fuel which has yielded 3,000 MW days, according to Goodlet still contains about 4 kgm. of plutonium. At present, metallurgical difficulties preclude the extraction of the toxic plutonium, mixing it with part of the depleted uranium, to form a fuel equivalent in fissionable atom content to the original fuel.

Thus, even in reactors known to be using natural uranium fuel, it is difficult to estimate the quantity of uranium likely to be used for replacement purposes.

Plowden (1957) gives an overall figure of one ton of natural uranium per MW of generating capacity for the initial charge. He considers this to be an average figure for all types of projected reactors whether they use natural or enriched fuels.

Goodlet's figures are used in an estimation of the uranium requirements of the British programme. For other calculations the more conservative estimate of Plowden is used (see Fig. 7).

Hagart (1957) gives generating costs at Calder Hall as follows:

Capital costs with 5% interest rate	= 0·4d.	per	Kilowatt	Hou
Running costs	= 0·3d.	"	"	"
Total costs	= 0·7d.	"	"	"
Value of plutonium	= <u>0·1d</u>.	"	"	"
∴ Nett Generating Cost	= <u>0·6d</u>.	"	"	"

<u>N.B.</u> Calder Hall is a base load station and therefore operates at a high load factor.

The generating costs of a modern British coal fired station, operating at a high load factor and having a thermal efficiency of 30%, is given by Beecham (1956) as 0·6d. per Kilowatt Hour (this figure was calculated on the basis of coal of 10,755 B.T.U. per lb. being delivered to the station at 65/- per ton).

The above comparison shows approximately equal generating costs for Calder Hall and coal fired stations in the United Kingdom. It is confidently anticipated by competent authorities that generating costs will be considerably less than those at Calder Hall.

The capital costs of the first three stations of the United Kingdom programme is about £35-40 million each, for a combined capacity of 900 MW (Goodlet, 1958). That is, about £125 per kilowatt of capacity, excluding the cost of the initial fuel charge. A coal station in Great Britain projected for the early 1960's, with a capacity of 1,100 MW is expected to cost £40 million to build (Iron and Coal Trades Rev., May 10 1957. p. 1105). That is, approximately £36 per kilowatt.

The high capital cost element in nuclear power stations is significant. The interest factor in relation to the high capital costs of nuclear power compared with thermal power will act as a deterrent to the introduction of nuclear power stations. This is especially important where power is supplied by private companies (as in the United States for instance) which generally require a higher interest rate as a return for capital invested.

The running cost of a nuclear power station is also significant. The cost of the uranium fuel elements accounts for most of the running cost. The actual uranium selling price could therefore influence whether an electricity authority decides to embark on a nuclear power programme or not. Also as Plowden (1957) indicates, the future trend of uranium prices will influence the decision on what type of reactors to be adopted in nuclear power stations. If it is anticipated that uranium will become more expensive, then those reactors which achieve a high "burn-up", despite increased capital and fuel processing costs, may be more economic. On the other hand, if the uranium can be obtained at low cost, reactors using natural uranium having low "burn-up" may be the most favourable.

Therefore the selling price will have an important effect on the civil demand for uranium.

Since Russia is reported to have abundant supplies of uranium to meet her own requirements and those of the Eastern

European satellites, they have not been included in this market survey.

The importance of nuclear energy derived from uranium does not lie in its lower cost, but in its availability for countries throughout the world that do not have sufficient supplies of indigenous fuel to meet their own power requirements. Obviously, the main field of application in the 1960's is in the highly industrialized areas of the world. That is, where conventionally generated power is relatively expensive; where large aggregates of capital are available, and where there are well developed grid systems of power reticulation.

The United Kingdom, for example, is a nett importer of coal to try and satisfy her power requirements which are increasing at such a rate that they are doubling every ten years. The 6000 MW programme will require about 12,000 tons of uranium by 1965. Although there is no announced programme for the period following 1965, Plowden (1957) stated it is probable that there will be about 20,000 MW of nuclear capacity installed by 1975, requiring between 5,000 and 10,000 tons of uranium per year in the 1970's.

Western Europe suffers a shortage of indigenous fuels similar to that of the United Kingdom, but has a much greater electricity demand. The Euratom countries (France, Germany, Belgium, Netherlands, Italy and Luxembourg) have proposed the installation of 15,000 MW by 1967. This will require an initial charge of approximately 15,000 tons of uranium.

According to Landsell (1958) two power stations totalling 165 MW are projected for installation in Sweden by 1963. There are also 6 small reactors for heating blocks of flats, projected for the years 1961-70.

The United States, although having vast resources of fossil fuel, is entering the nuclear power field. Davis and Roddis (members of the USAEC) estimated that 80,000 MW of nuclear capacity will be installed in the United States by 1975, and Dr. Libby (member USAEC) has estimated American civil uranium requirements at 20,000 to 30,000 tons of oxide per year by 1975 (Plowden, 1957).

According to the Chairman of the Japanese Atomic Energy Commission it is hoped to have 274 MW of atomic power capacity installed by 1962. He also envisaged a total capacity of 3,000 MW in Japan by 1970.

In most countries of the world the indigenous supplies of coal, oil and water power, although not fully exhausted, are limited. For these countries nuclear power is one of several possible means of power production. Perhaps Australia is typical of this vast potential nuclear power market; according to Pennycuick (1958) it was decided at a Symposium on Atomic Energy held in Sydney in June, 1958, that the introduction of nuclear power in Australia is best left until costs are reduced and a more economical use of the fuel made. Nuclear power costs, as stated earlier, are almost certain to decrease. Conventional fuels, as they become more scarce, will become more expensive. During the 1960's and 1970's nuclear power will become of increasing importance in these countries.

Since only those countries with military atomic programmes have uranium extraction and diffusion plants, many countries will be largely dependant on the few, for both the initial fuel investment and for the reprocessing of spent fuel. It is interesting to note that the present design of reactors all have capacities greater than 50 MW - suggesting that it is the minimum size at which costs are at all comparable with conventional generating methods. Therefore, as Landsell (1958) points out, atomic power progress will be slower in the less industrialised and less densely populated areas. The development of the so-called "package reactor" of 5-50 MW capacity may result in a much greater application of atomic power.

The projected "package reactors" will have an important application in remote industrial regions where the transport costs of convention fuels are extremely high. Mount Isa Mines Ltd. in N.W. Queensland is an example of such an application. The economics of a nuclear power station for Mount Isa were fully investigated by company engineers, but it was finally decided to construct a coal fired station of 30 MW generating capacity (according to MIMAG - Vol.II, No.1, 1958). The coal is railed some 800 miles from the company's coal mine on the Queensland coast and it is delivered to the power station at a cost of £A8 per ton. The high capital cost of the nuclear station would be the main deterrent to its use in these circumstances. Two reactors, each of which are capable of assuming the full load must be constructed, since the fuel elements in each have to be periodically replaced. The fuel elements would have to be transported from Great Britain in the absence of any uranium metal extraction plant in Australia. Also, unless rejected after use, the fuel elements would have to be sent to either Great Britain or the United States for processing and rejuvenation.

As the living standards of the more "backward" countries of the world are raised, they will require increasing amounts of power. In Asia, for example, 70% of the power requirements are supplied by muscle power (N.Z.B.S. Atoms for Peace Programme - 3 Aug. 1958). The coal reserves in India amount to approximately 100 tons per head of population and would last only 11 years at the American rate of per capita consumption (Goodlet, 1958). The construction of an atomic power station sets demands that strain the technical and financial resources of even the so-called "advanced" countries. For this reason it seems unlikely that the uranium demand from the technically "backward" countries will be significant in the forseeable future.

There are other uses of uranium as a fuel, probably the most important of which is in submarines. Nuclear energy is peculiarly suited to submarines for two reasons:

1. Power can be generated in a heat engine without the use of air or oxygen.
2. Fuel consumption is very low and therefore the range without refuelling is enormous.

The U.S.submarine "Nautilus", burning an enriched fuel, operated for 60,000 hours before the fuel was replaced (Hagart, 1957). Atomic powered submarines are being built by both the United States and Great Britain.

There is also the prospect of nuclear powered merchant ships. Large nuclear powered tankers designed to circumnavigate the Cape of Good Hope have been envisaged - design work in Britain is being done on such nuclear vessels. A 40,000 ton nuclear powered icebreaker was recently launched in Russia. It consumes approximately 5 ozs. of fuel per day (Keys, 1958). Two 40,000 ton tankers are being designed for Japan.

Although there are problems yet to be solved there is an increasing <u>marine</u> application of nuclear power, both for military and civil uses.

The feasibility of nuclear powered <u>locomotives</u> for transcontinental transport is being investigated. In Russia a locomotive has been designed to run between Vladivostock and Moscow on approximately 1 lb. of enriched fuel - compared with 600 tons of coal per trip for a coal fired locomotive.

Radioactive isotopes are used in medical, botanical, and metallurgical research. This demand is obviously small in comparison with use a fuel.

There appears to be a rapidly expanding application of nuclear power in the future. Balanced against the increasing

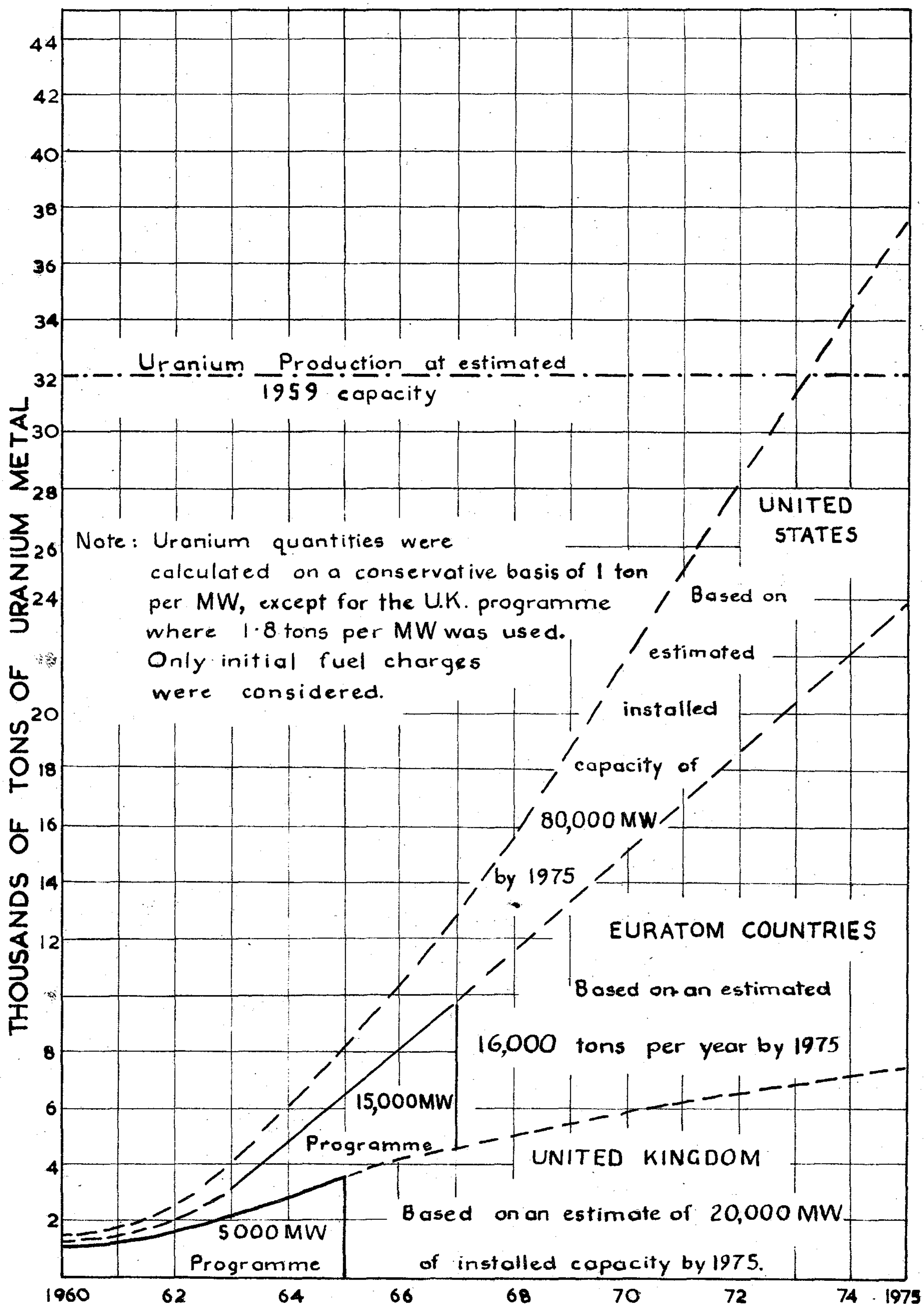


FIG 7. ESTIMATED URANIUM REQUIREMENTS FOR PROJECTED AND POTENTIAL NUCLEAR POWER PROGRAMMES UP TO 1975.

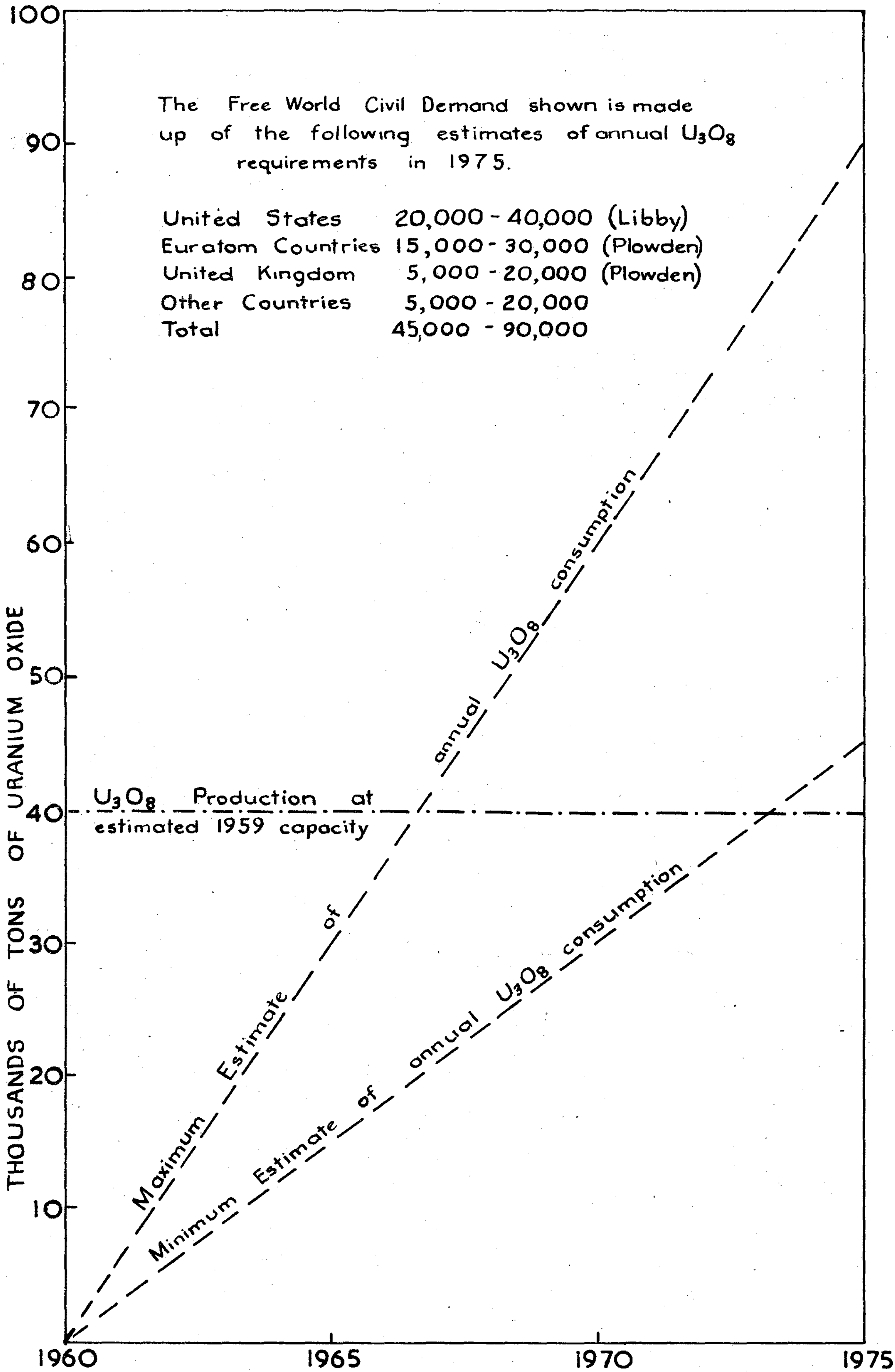


FIG 8: COMPARING POSSIBLE ANNUAL U O PRODUCTION AND CONSUMPTION UP TO 1975.

use of uranium is the use of thorium consuming breeder reactors and the ZETA process (Zero Energy Thermonuclear Assembly).

Thorium is not likely to completely replace uranium as a fuel but it will probably be used in competition with uranium. For instance, France and India, with large reserves of monazite are likely to use thorium reactors, should they prove practicable in the future. With widespread deposits of thorium already discovered, the effect of the use of thorium in breeder reactors will be to prevent the price of uranium rising above a certain value.

Thermo-nuclear fusion is reported to be well in the future. Various authorities have estimated that it will be 15 to 40 years before the problems of fusion are solved. Although at present it seems unlikely that the ZETA process will effect the uranium demand until about 1980, it is a potential hazard to the uranium mining industry.

Fig. 7 is a cumulative graph of the uranium required by possible major nuclear power programmes of the Free World shown in Table 4. Initial fuel charges only were considered where the conversion from MW of electricity to tons of uranium was made. With the steadily increasing uses in other countries, Fig. 7, should be regarded as a conservative estimate of initial fuel requirements.

Fig. 8 shows the large variance in estimates of annual Free World uranium requirements. The variance of 100% (between 45,000 and 90,000 tons of U^{308} in 1975) for the annual requirements is accounted for by the many unknown factors in the development of the nuclear power industry, some of which were briefly discussed earlier in this section.

The balance between estimated production and civil consumption is also indicated in the two graphs. It can be seen that it will not be until the 1970's that the rate of consumption for peaceful purposes, will be equal to the 1959 production rate of 40,000 tons of U_3O_8.

TABLE 4.

Possible Nuclear Power Demand of the Free World.

	By 1960	1960-65	1965-75
United Kingdom	1,000 MW	6,000 MW	15,000-20,000 MW
United States		5,000	75,000
Euratom Countries	150	15,000 (by 1967)	
Japan		274	3,000
Sweden	14 75*	75*	200*

* Denotes for heating purposes only.

6. Future Market Trends.

Since all uranium is sold by contract there is no established market and therefore no established market price, in the sense that there is in base metals for instance, where the London Metal Exchange daily price quotations reflect the market trends. All the uranium production of the Free World is contracted for by either the United States Atomic Energy Commission, the United Kingdom Atomic Energy Authority or the Combined Development Agency which is the official joint buying agency of these two countries.

Although contracts were negotiated individually with each producer, they were apparently similar in outline. For an example the salient points of the contracts negotiated between the C.D.A. and South African producers, (adapted from Hagart, 1957) are given below:

1. Each contract was for a period of ten years from the date of full production.
2. The Combined Development Agency provided the capital required for plant in the form of loan finance repayable with interest over the ten year period.

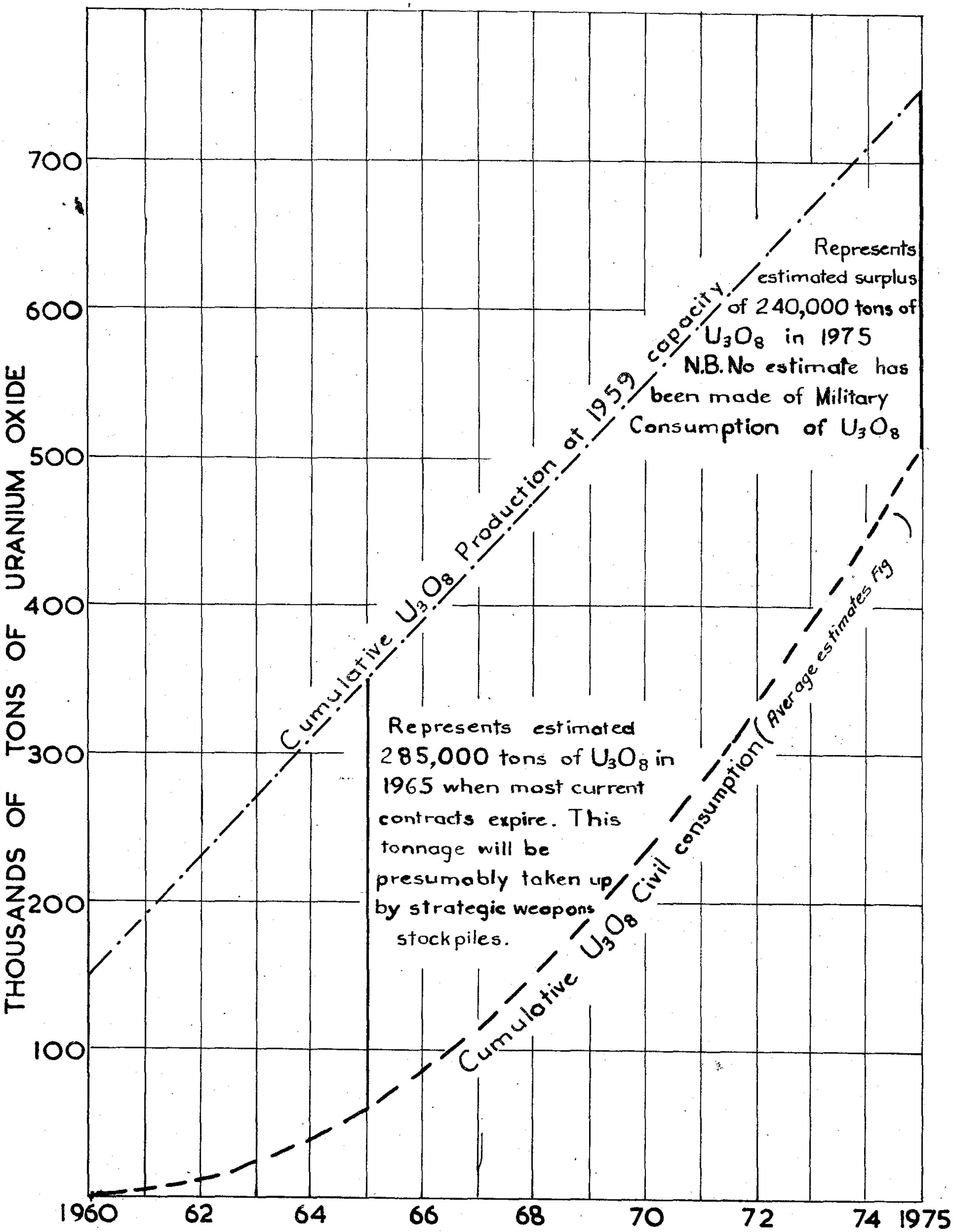


FIG 9: COMPARING ESTIMATES OF CUMULATIVE PRODUCTION AND CIVIL CONSUMPTION OF URANIUM UP TO 1975.

3. The price for the uranium oxide was based on a formula related to direct production costs, repayment of the loans plus interest, and a profit margin. The formula also provides an incentive to keep costs as low as possible by increasing the margin of profit when a reduction in cost is made.

4. An overall production target was set.

5. Provision was made for an adjustment of the price when there is an increase or decrease in costs, arising out of circumstances beyond control.

From a study of these general provisions two factors of significance are evident in the prediction of future price trends.

Firstly, the contracts are due to expire ten years after the commencement of production. This ten year period for the tenure of contracts appears quite general, except in the case of the Mary Kathleen mine whose contract with the United Kingdom Atomic Energy Authority is for 8 or 9 years from March 1959. Since the dates of the commencement of production of most plants were in the period 1953 to 1958 the contracts are due to terminate between 1963 and 1968.

Secondly, the capital costs of the mills and plants are calculated to be amortized during the tenure of the respective contracts. This means that after their contracts terminate during the middle 1960's, the established producers will be able to maintain profitable production at prices lower than those provided in the present contracts.

Fig. 9 shows that the Free World civil uranium demand is not anticipated to take up the full production by 1965. The uranium not required for peaceful purposes, estimated on Fig. 9 as 285,000 tons of oxide will presumably be absorbed by military stockpiling. It is reasonable to assume that by 1965 the major requirements of the military procurement programme will be fulfilled. The effect of stockpiling on the uranium market was discussed in the section on "Military Demand".

The United States is the largest uranium producer and the largest buyer of foreign concentrates in the Free World. Therefore as a guide to future market trends some recent developments in the United States are discussed below.

USAEC average contract prices for domestic and foreign impure mill concentrates are given in Table 5.

TABLE 5.

Average USAEC Contract Prices for Uranium Concentrates

U.S. Fiscal Year.	U.S. Domestic Concentrates in $ per lb. of U_3O_8.	Foreign Concentrates in $ per lb. of U_3O_8.
1956	11.60	10.90
1957	10.50	11.15
1958 (estimated)	9.60	11.15

In 1956 domestic ore-buying schedules were extended for a further 5 years until 1967. The base price of concentrates, which must meet certain specifications, in the period 1962-67, is given as $8.00 per pound of U_3O_8. Rather than indicating a continued military demand, this may have been an interim measure designed to bridge the gap between the curtailing of military demand and the increasing civil demand.

In October 1957, the USAEC announced:

"We have arrived at the point where it is no longer in the interests of the Government to expand production of uranium concentrates."

This meant that uranium deliveries under the USAEC's current domestic and foreign procurement commitments appeared adequate for military and nuclear power requirements for the ensuing few years. Since the foreign contracts were firm commitments, expected to reach a maximum tonnage of U_3O_8 in 1959, domestic production was restricted.

In 1957 the Combined Development Agency refused applications for expansion of South African production of uranium concentrates.

Although their ore reserves are not large, all Government sponsored exploration in the United States was terminated in 1957.

The USAEC has adopted a policy to encourage the gradual transition from a government controlled market to a commercial market as the industrial demand for uranium develops. It is likely that after the current contracts expire, the price of uranium concentrates, rather than be related to the costs of production etc., will be a direct reflection of the balance between supply and demand. Therefore, from the middle 1960's until the expected increasing nuclear power market takes up the surplus uranium in the 1970's, prices are likely to be depressed. The high cost, low grade mines (Radium Hill for example) will most likely prove uneconomic when their contracts expire. Any decrease in uranium price will tend to accelerate the introduction of nuclear generating units. During this period of depressed prices, the price will be very sensitive to military demand which is quite impossible to predict.

The actual level to which the price will descend, is also difficult to predict. A price of $6.00 per lb. of U_3O_8 is the lowest price at which most producers could operate. With the United States domestic price for the period 1962-67 set at $8.00 per lb. for a limited production, the "world price" could, therefore, be about $6.50 to $7.00 per lb. from 1965 to 1970. In the 1970's the price may reach $10.00 per lb. as demand increases. But the price is unlikely to rise above this, because of the competition from alternative forms of nuclear fuel.

7. Marketing Possibilities of the Buller Gorge Uranium.

The survey of the uranium market has been given in some detail because it is the inability to find a market that is retarding the development of the Buller Gorge uranium field.

As has been described, for the immediate future, the market appears to be satisfied by existing producers. However, a longer term outlook for the future of uranium is more encouraging despite the possible advent of thorium breeder reactors and the ZETA process.

At present and in the immediate future, the only substantial markets are the United States which is supplying the fuel for the Euratom programme, and the United Kingdom. The United States has restricted domestic production and is, therefore, not likely to purchase more foreign concentrates. The United Kingdom, obtaining its uranium requirements from Canada and Australia (Mary Kathleen production only) appears to have sufficient uranium to meet current demands for both military and civil purposes. Therefore, from a purely economic point of view, i.e. a supply-demand basis, New Zealand has little hope of negotiating an immediate marketing contract.

When uranium producers are in the open market in the middle 1960's it will be those producers who can mine and mill at lowest cost that will be able to continue to operate. The existing producers with their capital costs amortized, may sell at little more than working costs and have a great advantage over a mine commencing production at that time. If the Buller mine begins producing about 1965 it must be able to operate profitably with total costs less than $7.00 per lb. of U_3O_8. As the demand for nuclear fuel increases then price levels will tend to rise and the Buller uranium field could be more profitably exploited in the 1970's.

There is, however, one way in which New Zealand may find an immediate market. By reciprocal trade arrangements, Great Britain may find it to her advantage to purchase uranium concentrates from New Zealand. New Zealand can build up badly depleted sterling funds to buy the goods that Great Britain wants to export Without considering an increased military demand, this appears to be the only way in which New Zealand may find an immediate market.

The discussion of the uranium market, has been confined to mill concentrates, that is, the supply-demand trends of impure uranium oxide. This product requires a good deal more processing before it can be used as fuel in nuclear power reactors and it is the price of the nuclear fuel elements with which the ultimate market is concerned. At present the production of the uranium fuel elements is confined to Great Britain and the United States.

The increasing value of uranium with each stage of processing was given by Dickinson (1955) as follows:

			per lb.
	1.	U_3O_8 in ore	$ 3.00
	2.	U_3O_8 in concentrates (mechanical)	$ 7,50
*	3.	U_3O_8 in crude oxide (chemical)	$10.00 - 12.00
	4.	U_3O_8 in purified oxide (chemical)	$15.00
**	5.	U_3O_8 in purified feed (hexafluoride) to gaseous diffusion plants	$18.00 - 20.00
**	6.	U_3O_8 in metal feed (natural uranium) to reactor plants	$23.00 - 28.00

* Stage of processing at which uranium is sold by most producers.

** This refers to the <u>equivalent</u> U_3O_8 in the product.

The above, are general prices and according to Dickinson, do not represent official figures for any one mine or company.

By having an integrated production, the selling price of the product is increased from $10.00 - 12.00 to $23.00 - 28.00, i.e. an increase in price of 230%.

The advantages of being an integrated uranium producer are obvious. In this sense uranium is no different from other metals. For example, if the Taranaki ironsands were exported, rather than a steel industry being established in New Zealand, the loss of revenue to New Zealand would be enormous. If the uranium fuel elements could be produced at a cost competitive with uranium fuel elements from other sources, New Zealand uranium would be in a much stronger position on an open market than when selling the impure oxide. The capital cost of the plant required would be high. Sufficient uranium reserves in New Zealand would have to be proved to justify the plant, or alternatively the Buller mine production could be supplemented by importing uranium concentrates for further processing, from Australia for example. New Zealand could then supply the uranium fuel elements to the slowly developing market in the Pacific area.

The integrated production of uranium in New Zealand would necessitate large-scale, bold planning and would be a strain on the technical and financial resources of the country. However, such a project would give greater economic stability to New Zealand. More particularly, it will give a greater stability to the uranium mining industry when the possibilities of finding a market for the uranium concentrates are not encouraging.

The detailed requirements of the production of nuclear fuel elements are beyond the scope of this thesis, but it is at least worthy of some consideration in planning a uranium mining industry in New Zealand.

TABLE 6.

The Uranium Series

Element	Group	Isotope	Half Life	Principal Energies in MeV. Alpha	Beta	Gamma	% Total Activity Detected in practice Beta	Gamma
UI	URANIUM GROUP	U 238	$4\cdot5 \times 10^9$ yr.	4·18	-	-	approx. 43%	less than 1%
UX_1		Th 234	24·5 day	-	0·1, 0·2	0·09		
UX_2		Pa 234	1·14 min.	-	0·32, 1·5	0·78, 0·82		
UII		U 234	$2\cdot4 \times 10^5$ yr.	4·76	-	-		
Io		Th 230	$8\cdot3 \times 10^4$ yr.	4·68, 4·61	-	0·07		
Ra	RADIUM GROUP	Ra 226	1590 yr.	4·79, 4·61	0·09	-	approx. 57%	99 %
Rn		Em 222	3·825 day	5·49	-	-		
RaA		Po 218	3·05 min.	6·00	-	-		
RaB		Pb 214	26·8 min.	-	0·7	0·047, 0·35		
Ra C		Bi 214	19·7 min.	-	1·65, 3·17	1·76, 2·19		
RaC^1		Po 214	150 μ sec.	7·68	-	-		
RaD		Pb 210	22 yr.	-	0·026	0·05		
RaE		Pb 210	5·0 day	-	1·17	-		
RaF		Po 210	140 day	5·30	-	-		
RaG		Pb 206	Stable	-	-	-		

VII. RADIO-ACTIVITY DETECTION & RADIOMETRIC ASSAYING.

1. Radio-Active Decay.

The nucleus of the uranium atom disintegrates into daughter elements which in turn spontaneously disintegrate into other elements until the stable end product, an isotope of lead, is reached. The rate at which radioactive elements decay is given by the half life, which is the time required for the decay of half the atoms in a given amount. From Table 6 it can be seen that the half-lives of the members of the uranium decay series varies from several thousand million years for U^{238} to hundredths of a second for Po^{214}.

If undisturbed for long periods of geological time a uranium ore will reach a state of radio-active equilibrium. That is each member of the decay series is formed at exactly the same rate as that at which it decays. Theoretically, a uranium ore in equilibrium contains all the daughter elements in amounts inversely proportional to the ratio of their half-lives to that of the mother isotope (U^{238}).

Table 6 shows the energy in milli electrons volts (Mev) of the alpha, beta and gamma activities of each member of the uranium series.

2. Inequilibrium and Natural Leaching.

Inequilibrium occurs through the loss of one or more of the members of the decay series.

The gas radon, which is a member of the decay series, escapes readily from exposed uranium outcrops. It can also escape when the uranium minerals are finely dispersed, as is the case of coffinite in the Buller deposit, and where faulting and fracturing of the mineralized horizons are prevalent.

Uranium has two valency states U^{+4} and U^{+6}, The hexavalent UO_3 readily forms uranyl salts in acid solution according to the equation:

$$UO_3 + H_2SO_4 \rightleftharpoons UO_2 + SO_4^{=} + H_2O$$

These uranyl salts are very soluble in aqueous solutions. On the other hand UO_2 is relatively insoluble in non-oxidising acids.

Phair and Levine (1953) showed that unaltered primary uranium minerals are relatively insoluble in even concentrated solutions of H_2SO_4 at room temperatures. However the primary uranium minerals are readily susceptible to solution in dilute H_2SO_4 once they are partly <u>oxidised</u> by contact with oxygen, either in the air or dissolved in ground waters.

For leaching to take place then, the solution must be acidic, and the conditions oxidising. Natural acid solutions imply the presence of pyrite (or other sulphide minerals) in sufficient concentration, in or near the mineralized horizons, for rain water to become acidicenough to dissolve out the oxidised coffinite. Although pyrite is present in the Hawks Crag Breccia, it is argued that it is of insufficient concentration to be instrumental in causing large scale natural leaching. The high rainfall of the area is a factor which would encourage leaching. But again it is arguable that the steep slopes of the area, giving a swift run off, would allow insufficient time for leaching to take place. Little weathering of the uraniferous horizons appears to have taken place.

The writer has, however, found some evidence of relict pyrite structure or gossan in the radio-active horizons of the Buller; thus showing that, locally at least, conditions have been suitable for the oxidation and solution of pyrite. Also, in some places which are sheltered from the weather, the light coloured secondary uranium minerals are visible.

When geochemically testing the creeks of the Buller area for uranium, Wodzicki (1958) found that the uranium content of surface waters was highest during heavy continuous rainfall following a long period of drought. This certainly indicates that some leaching is taking place, but the extent and depth of surface leaching cannot be predicted with any reliability at this stage.

The extent of natural surface leaching of the uranium horizons is a most important factor in determining the reliability of outcrop sampling. If extensive leaching has taken place, then surface samples, even when chemically assayed, are no guide in the estimation of ore grades at depth.

A famous example of surface leaching is the huge Canadian Blind River uranium deposit where the ore is contained in a pyrite-bearing conglomerate. Early prospecting of the area with Geiger counters revealed strong radioactivity, but very low chemical assays for uranium and thorium were obtained. F. R. Joubin was of the opinion that the uranium had been leached out of the outcrops and the surface radioactivity was due mainly to the members of the radium group of the uranium series (see Table 6). In 1953 his theory was tested by some shallow drill holes. That Joubin's theory was correct is well illustrated by the fact that in 1958, only five years later, the Blind River uranium field has an estimated production of 12,000 tons of uranium oxide per year, and an ore expectancy of 500 million tons.

3. Radio-Activity Detectors.

Because the daughter elements in a sample in equilibrium are in fixed proportions to U^{238}, the activities of the uranium series may be considered as a quantitative determination of the uranium present.

Beta counters and the widely used simple gamma counters can be used for detecting radioactivity but they detect radioactivity from all sources of which the uranium series is only one. Thus they cannot distinguish between uranium and thorium and they also fail for assay purposes, if the ore is out of equilibrium.

Most of the naturally occurring radioactive elements belong to three major families; the parents elements being uranium ($_{92}U^{238}$ and $_{92}U^{235}$) and thorium ($_{90}Th^{232}$). The two uranium families always occur together in nature, and the ratio U^{235} to U^{238} is always constant and is equal to 1 : 139·2. Most of the world's known uranium ore bodies contain some thorium, but no thorium has been detected in the Buller Gorge deposit.

If, for example, the isotope Bi^{214} was removed from the ore, than a gamma radiation detector would give an abnormally low equivalent uranium assay, since the gamma radiation is emitted mainly by Bi^{214}. Therefore, for radiometric assays the ore must be in equilibrium. Inequilibrium, caused by the escape of radon, which is in the radium group supplying 99% of the detectable gamma radiations, is a common source of error in radiometric work.

A radioactive isotope of potassium, $_{19}K^{40}$ which comprises 0·012% of all natural potassium, emits gamma rays of relatively great penetrating power. Occasionally it may be the principal contributor to the radio-activity of a rock, although its effect on uranium is usually negligible.

Originating in outer space, those cosmic rays which survive the atmosphere have wave lengths similar to gamma rays. Because cosmic rays are not easily distinguishable from ordinary gamma rays, gamma ray detectors also receive cosmic impulses and therefore the total count has the "background count" subtracted from it to give the equivalent uranium assay.

Although normal radiometric analyses have the merit of great speed and cheapness, little reliance can be placed on them as accurate assays, because of the considerable errors to which they may be subject. This is especially true of surface samples because of inequilibrium factors.

4. The Equilibrium Method of Uranium Determination.

Eicholz et al. (1953), have devised a neat and ingenious radiometric method of determining the true U_3O_8 content of ore samples, regardless of the state of their radioactive equilibrium.

The method consists of the simultaneous measurement of the beta and gamma activities of an ore sample, thus giving an equivalent uranium assay due to the gamma count alone (U'_γ) and an equivalent uranium assay due to the beta count (U'). The actual uranium content of the ore can then be obtained by the equation of Eicholz, to whose work reference should be made for the derivation of the equation and a fuller discussion of the method. Briefly, let U be the actual uranium content of the sample, then according to Eicholz:

$$U = (1 + a)U'_\beta - U'_\gamma .$$

where $a = \frac{\text{beta count from the radium group of an ore in equilibrium}}{\text{beta count from the uranium " " " " " "}}$

The only assumption made is that the weathering or leaching of an ore sample does not disturb the $\frac{UI}{UX_2}$ from the value for an ore in equilibrium; thus the method can be applied to all but chemically purified ore samples.

The ideal equipment set-up is to have the sample centrally located between a Geiger tube which registers the beta particle activity, and a scintillation crystal which registers the gamma radiations. A diagram of the apparatus used at Rum Jungle (after Greaves (1957)) is shown in Fig. 10.

Although this equipment was not available at the Otago

School of Mines an end window $\beta-\gamma$ measuring Geiger tube and ratemeter were used for some radiometric assays of Buller Gorge ore using Eicholz's equilibrium method. To facilitate the measurement of the gamma radiations a close fitting aluminium plate lid was placed over the sample to screen the beta particles. The assays were carried out on finely crushed samples taken from the "A" horizon.

The instrument was calibrated for beta and gamma activity using standard samples of ·5%, 1%, 2% and 4% uranium respectively (obtained from the U.S.A.E.C.). These were assumed to be in radioactive equilibrium.

When the "background" count was determined, (this was done before and after each run of samples, and any changes assumed to be linear) a one gram sample of the ore to be tested was placed in the tray, the count recorded, and then the aluminium lid was placed over the sample and the process repeated. At least three runs were made on each sample. In this way beta and gamma counts were obtained for each determination and by reference to the standard calibration graphs the values of U'_β and U'_γ were interpolated.

It is readily admitted that the technique and equipment used were relatively crude. Since the beta and gamma counts are not made simultaneously, changes in "background" during the counting time are a large source of error. For instance, because of the low gamma counts recorded by the Geiger tube (1 to 10 counts per minute), the sample was left in the counter for periods of up to 1½ hours to obtain a statistically adequate number of radiations. During this long period, changes of "background" could readily result in errors of over 100% in the nett gamma count.

The ratio "a" as determined experimentally by Eicholz using aged natural pitchblende of known composition and pure uranium oxide, was used in the experiments. It is anticipated that "a" which indicates the relative contributions of the uranium and radium groups, may vary slightly with the different types of equipment used.

According to Eicholz: $a = \frac{\beta_R}{\beta_U} = \frac{57\%}{43\%} = 1 \cdot 30$

and the equation becomes $U = 2 \cdot 3U'_{\beta} - 1 \cdot 3U'_{\gamma}$

The results of the tests are shown in Table 7 and are compared with averaged chemical (fluorimetric) assays of the same samples.

TABLE 7.

Sample	U'_{γ}%	U'_{β}%	$U_{equil.}$%	$U_{chem.}$%
A100	0·25	0·39	0·57	0·554
A104	0·005	0·22	0·49	0·43
A106	0·175	0·20	0·23	0·262

Considering the crudeness of the technique used, the radiometric assays compare favourably with those determined by the fluorimeter.

Now it can be seen from Table 7 that U'_{γ} is not equal to U'_{β} in any of the samples; this shows that the samples are not in radioactive equilibrium. Further, in all samples U'_{β} is greater than U'_{γ} indicating a deficit in the radium series, which supplies 99% of the detectable gamma radiations. This is almost certainly due to radon losses. However this does not preclude leaching of the uranium, and it is the writer's opinion that both leaching and radon losses contribute to the inequilibrium of surface samples from the Buller.

The equilibrium method of U_3O_8 determination is used at Rum Jungle and Mary Kathleen uranium mines for routine grade control assays. It has the advantages over chemical analyses, of speed and of ease and cheapness with which the assays can

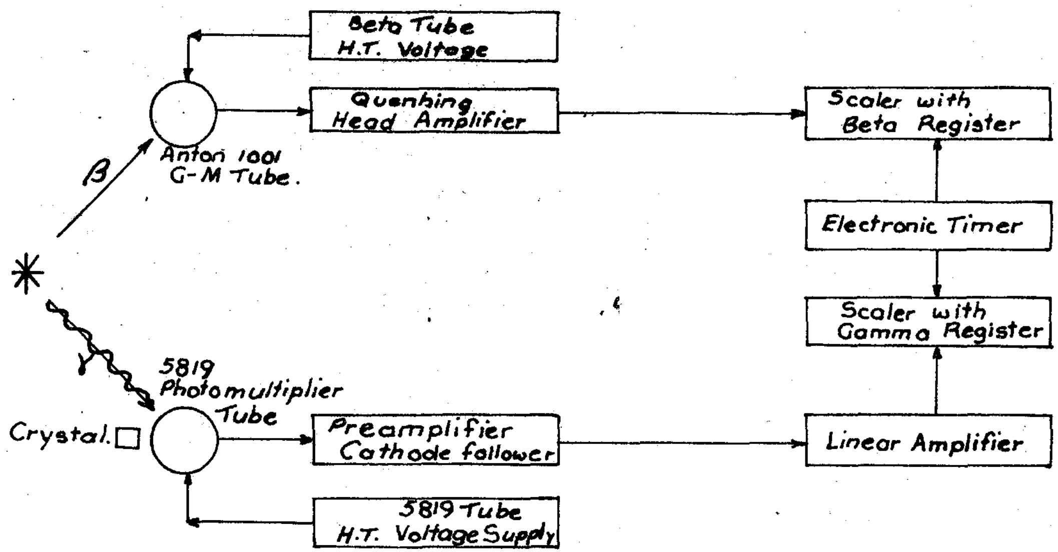


Fig 10 DIAGRAM OF EQUILIBRIUM UNIT (After Greaves, 1957)
Cost of equipment £A1,640.

be done. Also, compared with the skill in manipulation required for chemical assays, the radiometric equipment is simple to operate, once set up.

For the analysis of drill core and cuttings, outcrop and adit samples from the Buller it is recommended that the equilibrium method of Eicholz be used with the proper equipment (similar to that shown in Fig. 10).

VIII. URANIUM MINING HEALTH HAZARD.

1. Uranium Poisoning.

Uranium is one of the most toxic elements, chemically more toxic than either arsenic or mercury. However, it is only absorbed into the body with difficulty (even when swallowed only traces are absorbed). Uranium that penetrates to the kidney is a source of injury since it kills cells by blocking carbohydrate metabolism. Much is known about uranium poisoning, but in all uranium plants, mills and mines, cases of even mild uranium poisoning are extremely rare.

2. Biological Effect of Radon and its Daughter Elements.

The radiation hazard in mining uranium ores comes from the radioactive gas, radon and its daughter elements RaA, RaB, Ra and RaC'.

A reference to Table 6 shows that the gas radon has a half-life of 3·8 days, while its immediate daughter products have half-lives of much shorter duration. Table 6 also shows that RaA, RaC' and radon give high energy alpha particle emmissions. The relative damaging effects of gamma rays and beta particles are very small compared with high energy alpha particle emmissions which have a high specific ionization. These interact readily with body cells, especially the soft tissue of the lung. Thus, the potential hazard to the lung arises mainly from the alpha particles from radon, RaA and RaC'.

The radiation received by the tissues of a mine worker is derived from two sources:

1. Radon gas, which is inhaled with the air of the mine.
2. The radioactive daughter products of radon which remain suspended in the mine atmosphere as fine particles.

with its longer half-life much of the inhaled radon is expelled in breathing before it can decay. The radon does, however, emit alpha particles in an amount proportional to the concentration of the radon in the air, and continues to decay in the lungs. Also some radon enters the bloodstream. The solid decay products of radon adhere to the large numbers of dust particles and water droplets usually present in the mine atmosphere. A portion of this dust is retained in the lungs during breathing. In this way a percentage, measured as being between 25% and 75% of the decay products inhaled, is retained in the lungs where radioactive decay continues.

Experimental studies by Bale and Shapiro (1956) suggested that the primary hazard from breathing air containing radon was from the accompanying airborne daughter products. Making certain assumptions Miller et al (1956) have shown that the ratio of the radiation dose derived from inhaled radon and its decay in the lungs, to the radiation dose produced by decay products breathed from the air, is 5 per cent. That is, if the solid decay products were filtered from the air, the average radiation dose to the lungs due to radon and its daughter elements would be reduced to 5% of the value for unfiltered air.

3. Maximum Tolerance.

A high percentage of deaths due to lung cancer (estimated as 30 to 50% of all deaths) among miners in the Schneeberg radium mining region of South Germany and the adjoining Joachimstahl of Czechoslovakia have been commonly attributed to the radon present in the mines. After analysing radon measurements made in these mines (Evans and Goodman (1940)) concluded that the average concentration was about 10×10^{-10} curies per litre. Evans suggested that the maximum permissible level for radon ought not to exceed 1% of this value, i.e.

$0{\cdot}1 \times 10^{-10}$ curies per litre. This figure has been widely accepted by the luminous indicator and watch dials industry. In France the maximum tolerable concentration of radon in mines is set at 1×10^{-10} curies per litre, while the general standard set in most States of the U.S. is 3×10^{-10} curies per litre of air.

Although the importance of the concentration of radon daughter products in the atmosphere is now recognised by most workers, sufficient biological data are not available to determine finally a maximum permissable concentration for radon daughter products. Officers of the U.S. Dept. of Health (1957) suggested for uranium mines, a working level of $1{\cdot}3 \times 10^{5}$ milli-electron volts (Mev) of potential alpha energy per litre, for the radon daughter products of RaA, RaB and RaC. This amount of energy is released by the decay of 1×10^{-10} curies per litre, of each of these elements down to RaD. It is maintained that this level (in the light of present information) is reasonably safe.

The present New Zealand legislation regarding the ventilation of underground metal mines provides that ventilation ..."shall be at the rate of not less than 150 cubic feet per minute for each man employed ... and distributed so that at least 150 cu. ft. of air per min. be supplied at every working place, for each man employed." Clause (e), section 94, Mining Regulations, 1935. That this quantity of air is insufficient for safe working in uranium mines is evident from the following additional recommendations made by the U.S. Dept. of Health in 1957, regarding ventilation requirements:

"1. Each pair of men mining in ore should be provided (with) no less than 1,000 cu. ft. of fresh air per min., discharged from a tube outlet, located not more than 30 feet from the face.

2. In drifts, the quantity of airflow should be calculated to produce a velocity of not less than 30 feet per minute."

It is obvious that the provision of a certain arbitrarily determined minimum quantity of air for each underground worker, can only be used as a guide in ventilation planning. In such a regulation no account is taken of the rate of radiation emanation from the ore - this is the basic factor in the ventilation of uranium mines.

It is proposed that the working level of $1 \cdot 3 \times 10^5$ Mev of alpha particle energy per litre of air, recommended by the U.S. Dept. of Health in 1957, and accepted as a standard by the USAEC, be taken as the maximum tolerance of radiation permissible if uranium mining is undertaken in New Zealand. New Regulations would be necessary to give effect to this.

4. Sampling Techniques.

In order to have some degree of control on the radiation hazard it is necessary to assess the concentration of radon and its daughter elements in the mine atmosphere and thus determine the emanation rate.

There are several ways in which the concentration of radon can be measured; in laboratory techniques it is usually the ionization effect due to alpha particles that is measured. Several different types of instrument apparently give satisfactory results. There follows a brief description of one such instrument, the radon scintillation cell, which is suitable for use in the field. A 125 c.c. flask, coated on the inside with zinc sulphide phosphor, is evacuated by connecting to a vacuum pump, and a sample of the air to be tested is drawn into it. After a sufficient time interval has passed for equilibrium to be reached, the scintillations are counted using a standard photo-multiplier tube and scaler combination. It is reliable for samples containing at least $0 \cdot 5 \times 10^{-10}$ curies of radon per litre of air.

Another method of sampling is to pass air through a cartridge containing radioactive carbon, which absorbs the radon.

Yourt (1955) considered that for routine ventilation control requirements, the measurement of the radon daughter element's concentration were too slow and the equipment not suitable for underground use. However, since it has been found that the radiation dose to the respiratory system is largely due to the daughter elements of radon being retained in the lungs, it is important to ascertain the amounts of these elements in the mine atmosphere.

The U.S. Dept. of Health recommend the following field method for measuring the radon daughter products. Using an air pump a known quantity of mine air is drawn through a filter paper; molecular filters are considered the best, because they retain the very small dust particles. In a non-radioactive area, measurements of the radioactivity of the samples are made by a device which records the alpha activities of the samples. In this way the alpha disintegrations per minute can be measured. Using standard equations developed for this purpose the total energy released due to alpha particles can be calculated in Mev per litre. The total alpha particle energy released is calculated to obviate the need for determining the respective amounts of RaA, RaB, RaC and RaC'.

5. Control.

Radon enters the mine atmosphere by diffusion from exposed ore. The concentration of radon in the atmosphere is thus determined by the surface area of exposed ore, the rate of emanation from the ore and the rate of removal by ventilation.

The liberated radon decays according to its half-life, and some is removed by the outgoing air stream. The daughter

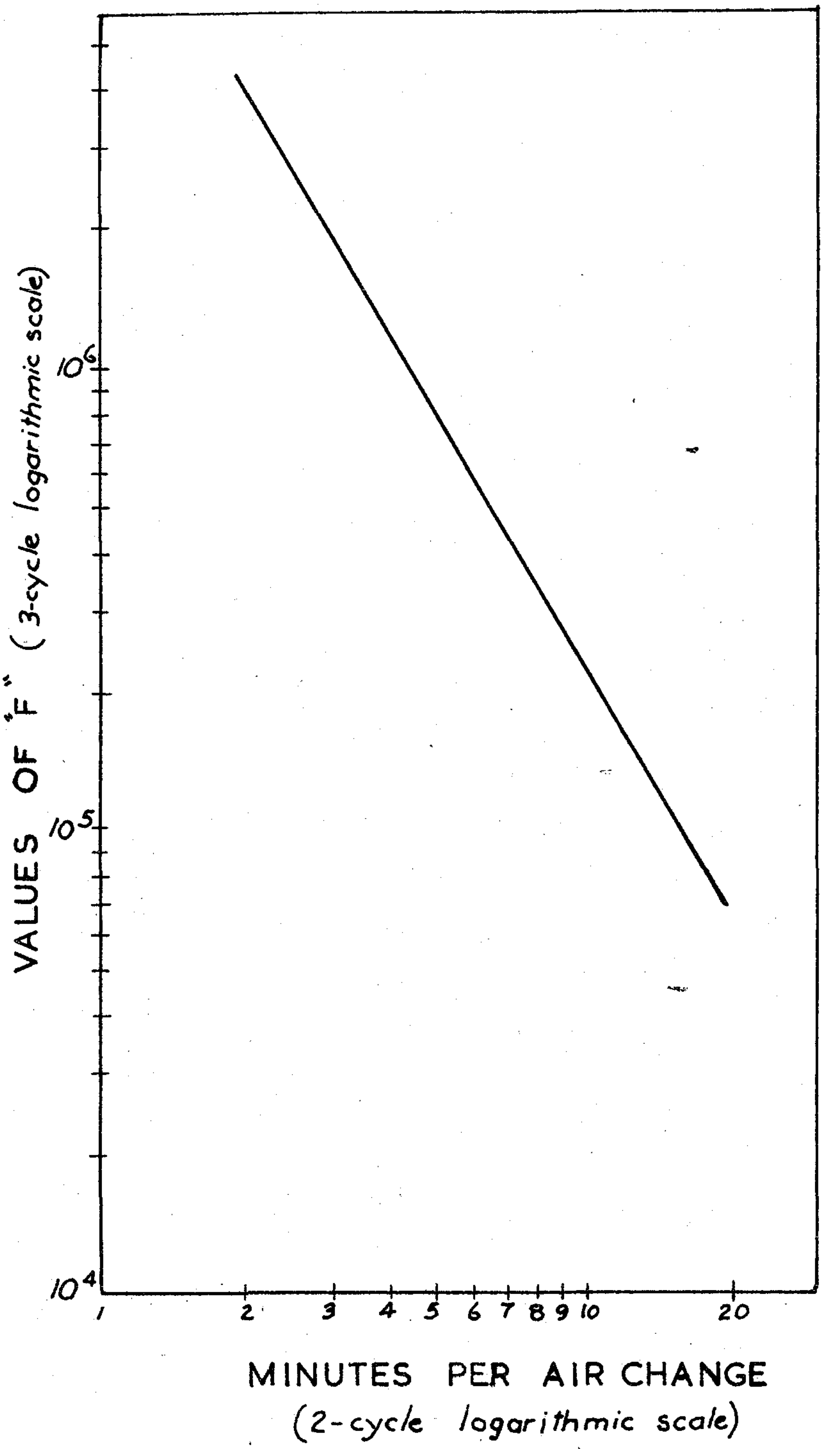


Fig. 11: (After U.S. Dept. Health.)

elements are formed in the air through decay of the atmospheric radon, and because of their short half-lives they will tend to reach equilibrium. Since it requires about three hours for equilibrium to be reached in this series, the atmospheric concentrations of these elements are affected readily by ventilation. This was proved by Tsivoglou (1956) who conducted experiments in a section of a mine on the Colorado Plateau. He showed that at the highest ventilation rate (one complete air change every three minutes) the atmospheric radon concentration was reduced from the unventilated condition by a factor of 200; the concentrations of RaA, RaB, and RaC were reduced by factors of 1000, 2500 and 2500 respectively. The actual radiation dose was only 7 per cent. of the dose calculated on the assumption that all the daughter elements were present in equilibrium concentrations.

Adequate ventilation of all working places with uncontaminated air is the most effective means of reducing the concentration of radon and its decay products.

In designing the mine ventilation system it is desirable to calculate the air quantities required to dilute the radiations to below the standard set. The determination of the emanation rate is, therefore, necessary.

A relatively accurate method is given by the U.S. Dept. of Health:

Let R = E.F.

where R = emanation rate in atoms per minute per 1,000 cubic feet

E = the measured daughter product concentration in Mev per litre.

F = a factor determined by the ventilation (F also includes factors which make the equation dimensionally correct)

E is measured, by the method described earlier, in the section on "Sampling Techniques." Fig. 11 is a graph (drawn up by the U.S. Dept. of Health) in which the values of F are plotted against different ventilation rates. From this graph the

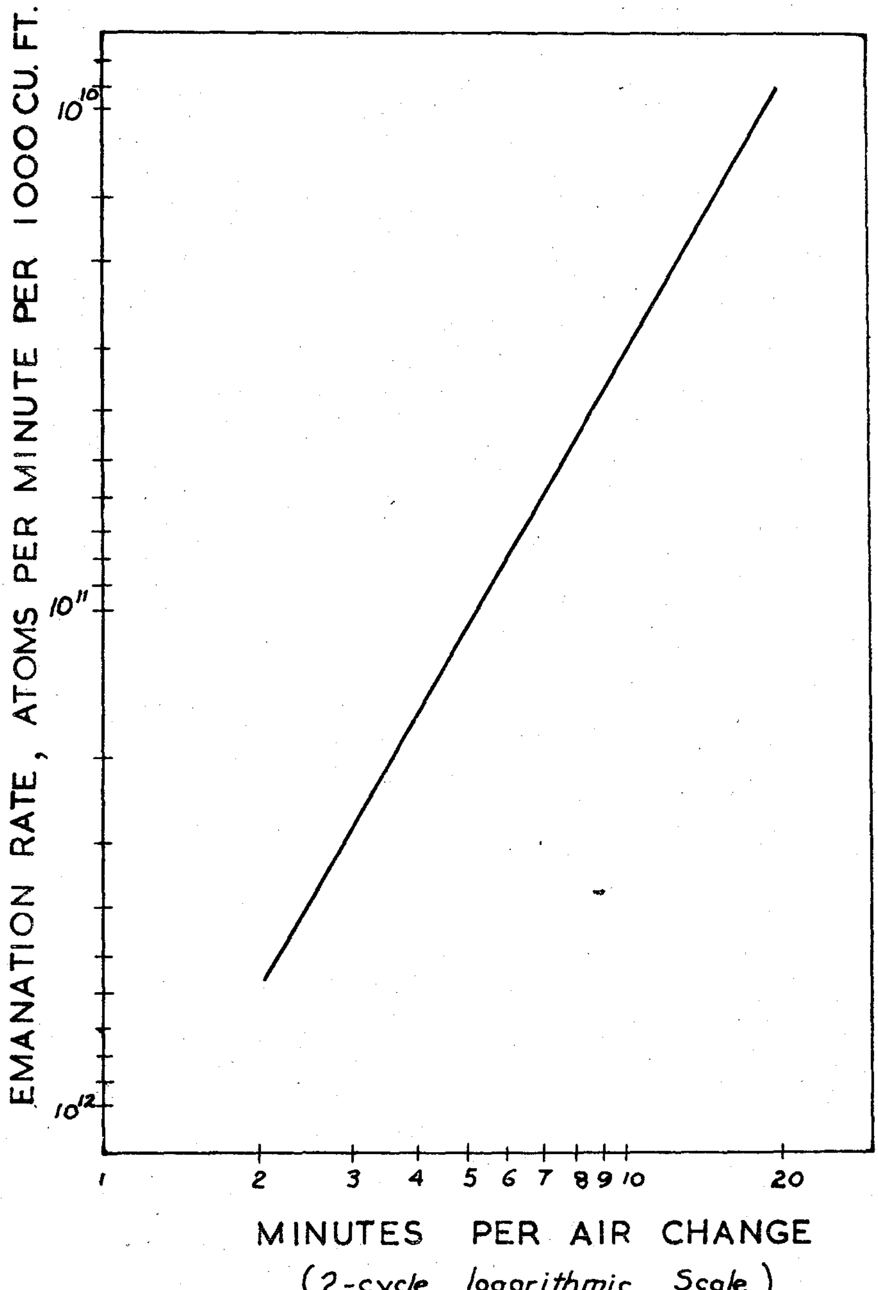


Fig 12: (After U.S. Dept. Health)

appropriate value for F can be plotted and hence R calculated. Now, with the emanation rate known, and the maximum alpha daughter product activity set at $1 \cdot 3 \times 10^5$ Mev, it is possible to calculate the ventilation rate necessary to reduce the alpha activity below that level. This calculation has been done for various emanation rates in Fig. 12 (drawn up by the U.S. Dept. of Health) which shows how the ventilation rate necessary to keep the maximum daughter product activity below $1 \cdot 3 \times 10^5$ Mev per litre, increases as the emanation rate increases.

In this method non-equilibrium conditions are adjusted but consideration is given only to the reduction in radon daughter element concentration by means of increased ventilation rate and radioactive decay. It also assumes the introduction of uncontaminated air for ventilation but does give a reasonably accurate estimate of the ventilation requirements for each section of a mine.

The control of dust is particularly important in reducing the inhalation hazard of RaA, RaB, RaC and RaC' which are attracted and held to dust particles of electrostatic forces. Wet drilling and wet handling of the ore have long been common practices for dust suppression as a control for silicosis.

In a ventilation survey of some Canadian uranium mines, Yourt (1956), found nearly all the RaA and RaC' collected was associated with dust particles less than one micron in diameter, But where the dust concentrations were high, the alpha activity was apparently associated with coarser dust fractions.

In extensive workings, ventilation of all working places with uncontaminated air may be probigitively expensive and practically impossible. Investigations have been made in the U.S. into the use of air cleaners as a means of control of the inhalation hazard. Some of the daughter elements are removed by recirculating the air through an air filtering device. The concentration of radon gas will tend to build up during this recirculation. A combination of recirculation with air filters and fresh air intake should, therefore, provide a practical and effective means of reducing the inhalation hazard of both radon and its daughter products to within the acceptable limits. This would be applicable, for instance, along a main "in-ore" haulage level, or a long-wall face, where the concentration of both dust and radon is high. The provision of adequate filtration plants is particularly important at ore transfer points, i.e. ore passes, tipples, draw points, etc.

Since the concentration of radon in the atmosphere is dependant on the surface area of the exposed ore, the following practices were considered when planning the method of mining suggested for the Buller Gorge:

1. Large fragmentation from blasting.
2. The rapid removal of ore once it has been mined.
3. The use of air-tight stoppings in closing off old workings.
4. A ventilation system which delivers relatively fresh air to the working faces and avoids lone "in-ore" airways. This can be done by working the mine in panels or sections well insulated from each other.

Slightly soluble in water radon can travel long distances in the water, to be released when the pressure on the water is lowered. Thus it is good practice to pipe the water (i.e. in closed conduit) outside the mine, as soon as it appears.

It is worthy of note that in France some experiments have been conducted on covering the ore, and studying the effect on radon release. Oil in which radon dissolves readily, is sprayed on the ore and is successful in delaying radon diffusion (Jamnet and Fradel 1956).

Finally, it can be stated that the provisions for ventilation and dust control is most well managed metal mines (especially where there are strong Trade Unions) have apparently proved sufficient for uranium mines.

IX. THE ESTIMATION OF ORE RESERVES.

1. Introduction.

The estimation of the grade and tonnage of ore is the key factor among those of capital outlay, selling price of the product, and likely working costs, which must be assessed before deciding whether the deposit is an economic proposition or not.

Proving the deposit is an essential process which must be designed so that expenditure is balanced against risk. From the time the discovery of the radioactive minerals is made known, each stage of development should justify as far as possible, the cost of the next consecutive stage. Thus the estimation of the deposit may be resolved into a number of stages, each of which is accompanied by the successive diminution of risk, until a stage is reached when capital can be attracted, with a fair chance of success.

In this way, and in this way only, can the inherent financial risks of mining be rendered comparable with those of normal business enterprises. This point is further emphasised by the fact that the large capital outlay necessary for the establishment of a mine and mill, is a prepaid cost. Also, with the significant number of financial failures in mining, involving the loss of millions of pounds, the risk capital required for the estimation of a mineral deposit, is difficult to raise, unless the proving work is done carefully and the chance of ultimate financial success is increased with each additional expenditure on exploration.

Once embarked on any one stage of proving, that stage must be worked to completion before any decision is made concerning the feasibility of subsequent stages. For instance, the assay information gained from one exploratory hole by itself is negligible. It is a fundamental error of sampling theory to assume that one drill hole is representative of the whole deposit. If one hole is drilled, then the mining company is committed to drilling a number of holes, in a scout drilling pattern, for example. The existence of the deposit at depth can be ascertained in this way. Similarly, if it is decided to sample the surface outcrops then all outcrops must be systematically sampled, not merely a few of the more accessible outcrops.

In the proposed method of estimation of the ore reserves, use is made of the science of statistics. The sampling programme is designed so that the principles of statistics and the theory of probability can be applied. The use of statistics in the estimation of ore reserves has met with increasing success during the past decade, and it has been built up on a solid practical foundation. It must, of course, be applied with due care and prudence, but it is a powerful instrument which enables the maximum amount of information to be obtained from sampling data.

2. Sampling Theory.

Sampling is essentially the process of inferring something about a whole from an examination of part of the whole: the hypothesis being that if enough small portions of the whole, properly spaced, are taken, their average value will approximate that of the whole. Statistics is a mathematically exact method of drawing general conclusions from fragmentary data. The one is, therefore, dependant on the other.

Although mining engineers have long recognised the difficulties and precautions necessary in sampling, it was not until 1947 when Sichel published the results of his work, that any general acceptance of the science of statistics came from the mining industry. It has since been proved, especially in the Witwatersrand and Orange Free State gold fields, that statistical analysis of sampling data is an invaluable aid in the estimation of reserves.

The science of statistics has been built around the theory of random sampling, which means that all particles in the ore deposit must have an equal chance of being selected in the sample. This condition can be approximated to, in the estimation of ore reserves, by an arbitrary, systematic and regular spacing of samples. There is nothing in the method of random selection to ensure that a representative sample is produced, but its virtue lies in what has often been described as its fairness.

It will be seen in the following paragraphs that the design of the sampling programme is such that all samples taken are regularly spaced. In doing this, the principle of random sampling is adhered to, as far as is practicable.

It is possible, however, that ore values may be distributed in such a way that they are changing in a regular and continuous manner over the region samples. In this case a set of systematically spaced cores will not give a random selection of all possible values, sampling results will be biassed, and a check on this occurring must be incorporated in the design of the sampling programme.

3. Method of Sampling.

The initial work of interpreting the geological structure from the surface geology has already been accomplished.

The first stage in the estimation of tonnage and grade of ore in the deposit is to systematically sample the known outcrops. This is obviously the cheapest and quickest method of obtaining samples.

TABLE 8.

Assay Results of Chip Samples from "A" Horizon

Locality	Radio-metric %	Uranium Oxide (U_3O_8) Chemical	Width Ins.	Remarks.
A.100	0.005	0.01	24	Hanging wall.
A.100	0.005	0.01	24	" "
A.100	0.06	0.055	24	Ore band
A.100	0.02	0.02	24	Foot wall
A.102	0.005	0.01	24	Hanging Wall
A.102	0.51	0.505	32	Ore band
A.102	0.02	0.02	18	Footwall
A.103	0.075	0.08	18	Ore band
A.104	0.10	0.10	24	Ore band
A.105	0.36	0.36	24	Ore band
A.106	0.04	0.04	12	Ore band

(By courtesy N.Z. Mines Dept.)

The "A" horizon has been sampled both by the chip and the channel methods of sample cutting. The assay results of the chip samples are shown in Table 8 and the location of the samples can be seen from Fig. 4. A significant feature of the assay results shown, is their variability - even over the same outcrop. The assays from the channel samples were inaccurate since it has been found that insufficient care was taken in both collecting and assaying the fine particles in the samples.

As discussed in a previous section, surface samples, even when carefully taken, are of dubious value in assessing ore grade because of natural leaching of the uranium. Now the immediate object of surface sampling is to assess whether or not the next stage in the estimation of the deposit is justified. Where the possibility of the alteration of values by leaching exists, another risk factor of indeterminate value is introduced. The elimination of this factor requires that the samples be taken from beneath the zone of leaching. This means that the outcrop should be blasted away until there is no mineralogical evidence of surface oxidation or weathering - a depth of 2-5 feet should prove sufficient on the steeper hill faces.

Because of the variability in values, a large or bulk sample should be taken at each location. The sample can be taken by the judicious use of explosives, with a tarpaulin covering on a rough timber stage to retain the sample. A sample size of about one ton i.e. about 12 cubic feet of the ore in situ should be the aim. This sample is then crushed, using a small petrol or hand operated crusher, to about -1 inch, care being taken to retain all the fines. The crushed sample is coned and quartered at the outcrop to finally yield two 40 lb. samples which can be packed out to the road.

Accompanying each sample, should be a label with outcrop serial number on it. To avoid any bias by the assayer to whom the sample is sent, no other information should be included with the samples. It is a good check on the accuracy of the assays to make say, every tenth sample a "blank" - i.e. of rock known to contain no uranium.

The sampling process outlined above will require patience and care, but is considered necessary in order to obtain an unbiassed sample.

A plan of the proposed sample locations is shown on Fig. 4 -- the interval between samples is four chains. It may be impossible to adhere strictly to this pattern because of inaccessibility, but minor differences in the distance between sampling points will have a negligible effect.

If the assays of the surface samples are favourable then it becomes possible to recommend that the next stage of proving the deposit be undertaken.

The second and subsequent stages of exploration are concerned with testing for the continuation of the mineralization at depth. There are two main approaches to the estimation of the ore at depth by drilling.

The first method consists of drilling on a close pattern grid (200 feet centres for example), beginning the grid pattern at the previously sampled outcrops and gradually extending it outwards. Small prospecting adits may also be driven in the mineralized horizons. The grid is extended until either the mineralization peters out, or sufficient ore reserves are established to amortize the capital cost of the mine and mill. Further ore reserves are established by underground development as the deposit is opened up. The method has the merit of few "blank" holes being drilled, but no overall knowledge of the subsurface geology is obtained. A high risk factor must be accepted because of the uncertainty of the ore expectancy.

The second method of approach is that which is proposed for the Buller deposit. The drilling programme is divided into two stages and has the advantage of being more systematically designed than the method described above. Scout pattern drilling on 1000 feet centres constitutes stage two of the exploration and development programme. The proposed location of the holes and their order of drilling is shown in Fig. 23. The grid has been oriented in such a way as to begin drilling near the "A" horizon and avoid as far as possible, siting holes on the highest ground. A drillhole spacing of 1000 feet was chosen, since it appeared to provide the optimum combination of cost and satisfactory exploratory results. The sixteen holes constituting the scout pattern, at an average depth of 800 feet, entail approximately 12,800 feet of drilling. By first drilling a scout pattern the whole area is covered, inadequately for ore estimation purposes, but the area is at least randomly sampled. The purpose of the scout drilling pattern is to ascertain the probability of mineralization occurring at depth. Another important use to be made of the information gained from scout drilling, is in the structural interpretation of the area. By considering results from any ore intersections together with other geological information obtained from the drill core analyses, the geologist can indicate those areas (if there are any) which are worthy of further investigation. Drilling should be continued in each scout hole until the geologist is satisfied that sufficient information has been obtained and there is no chance of any mineralized horizons occurring at greater depths. If a scout hole does not intersect ore, it cannot be assumed that there is no ore within the 1,000 x 1,000 square feet area of influence of the hole, conversely it is wrong to arbitrarily assume that the ore is distributed over the 1,000 x 1,000 square feet area of influence. But the scout pattern does indicate where the chances of subsequently proving ore are greater; thus always balancing risk against expenditure.

If any of the scout holes intersect uranium horizons, then those areas are marked for close pattern drilling, which constitutes stage three of the proving programme. The design of the third stage is dependant entirely upon the results obtained from scout pattern drilling. Holes may be necessary as close as 100 feet apart, depending on the statistical distribution of values obtained from the holes. The trends of the ore distributions are obtained from the close drilling pattern.

A check on the distribution of uranium ore values can be made if considered necessary. This can be done by drilling holes in the centres of the close grid patterns at 45 deg. to the vertical, directed along the ore trends.

Prospecting adits should not be considered until the third stage of the proving programme because geological information obtained from them would not extend over the full depth of the favourable rock type. However, in the third stage of the exploration programme the economics of prospecting adits should be investigated. Adits are more economic than surface drill holes when the cost of proving a particular block of ore by short drill holes from the adit is less than proving the same block of ore, with a similar accuracy, by surface drill holes. It is obvious that adits are likely to be the cheaper method of proving if the mineralized horizons are concentrated in a short vertical distance and do not occur sporadically throughout the Tiroroa Facies of the Hawks Crag Breccia.

Adits should be driven at a slight grade (1 in 100) to facilitate drainage and haulage. The cost of driving a 7 ft. x 5 ft. prospecting adit is estimated at £18 per foot. When the third stage has been completed i.e. when all areas indicated to be favourable for ore occurrence have been sampled, the total ore reserves can be calculated with an accuracy which can be determined by statistical methods. Knowing the ore reserves the mine can be designed to produce at the optimum output and lowest cost per ton of ore. This is most important for marginal grade deposits.

4. Statistical Analysis of Samples.

Throughout the discussion on sampling the need to balance risk against expenditure has been stressed. With the aid of statistics, this important principle can be carried on in the calculation of ore reserves from sample results. Using statistics, not only is it possible to estimate the average ore grade, but the pattern of the distribution of ore values about that mean can be ascertained. Further, on the theory of probability, the confidence which may be placed in the estimate of the average grade of ore can be measured. In other words a precise measure of the risk involved in accepting the mean of the samples as being the mean of the orebody represented by the samples, can be made. For example, Krige (1952) statistically analysed the results from 91 drill-holes in the Orange Free State gold field. He estimated the mean ore grade as 411 inch-dwts. and showed that there was an estimated chance of only one in 100 of the actual mean value being less than 239 inch-dwts., and a similar chance of its exceeding 750 inch-dwts.

The importance of the statistical approach to the estimation of ore reserves, may be judged by the following statement made by Luttrell-West (1952):

> "I have personally been associated with the problem of assessing the potential value of mining properties from boreholes and development exposures, and I can assure you that the recent developments in statistical research on this subject are not only valuable in the academic field, but also provide the first really consistent practical method of handling this problem .....
>
> For those who have to make decisions on financial commitments, it is very important to be able to judge and to have suitable guides to judge the degree of confidence in each factor used."

The primary consideration in the statistical analysis of a number of sample values, is the frequency distribution. All the assay results, regardless of the drill holes from which the samples were taken, are grouped into suitable equal-size grade intervals. The sample results can be represented graphically by plotting the limits of the range of each grade interval as abscissae, and on each grade interval as a base, a rectangle can be constructed with area in direct proportion to the frequency of occurrence of assay

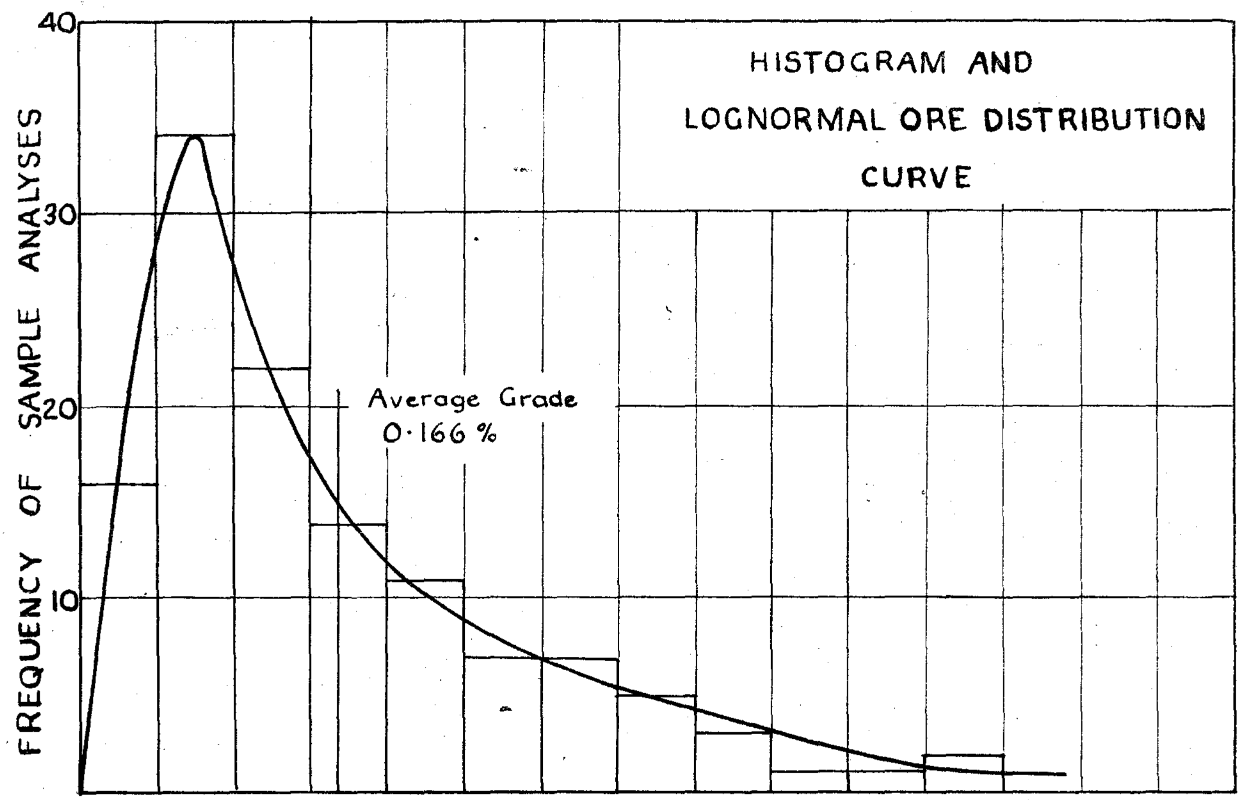
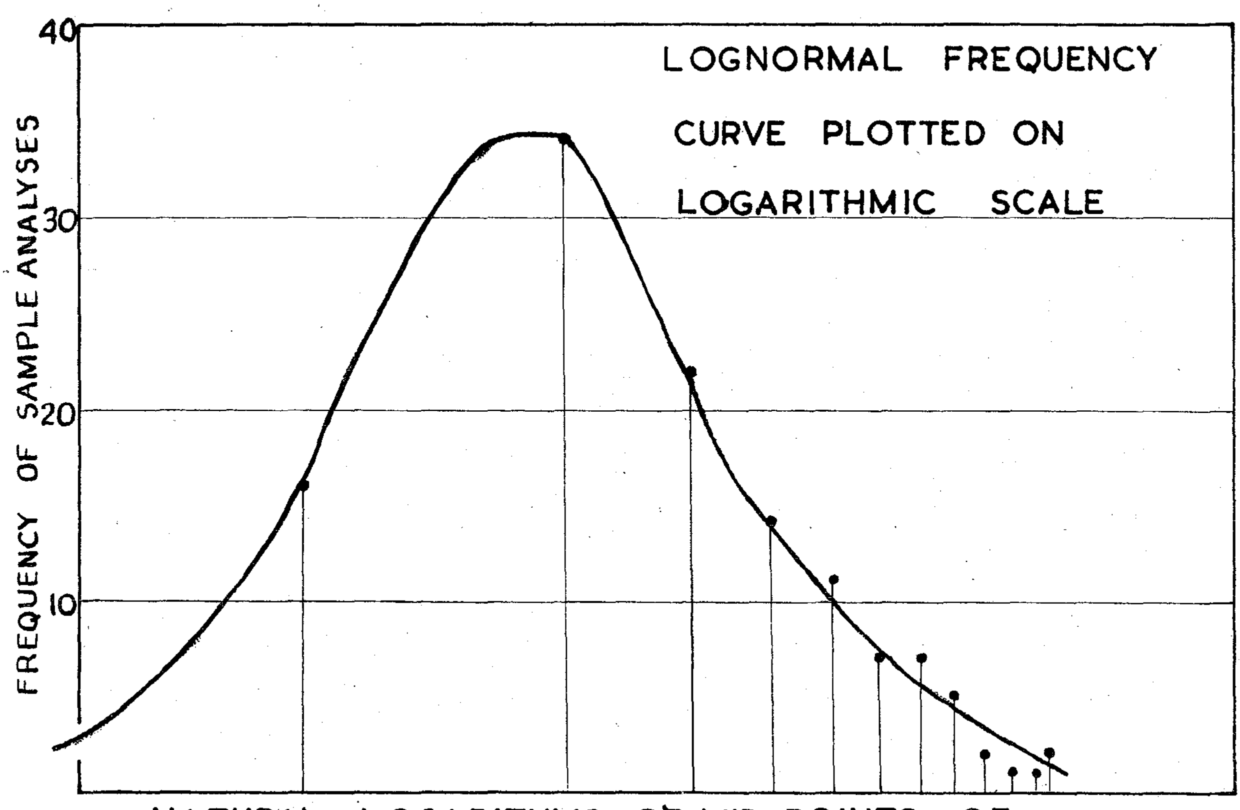


FIG 13: ORE GRADE % U_3O_8



NATURAL LOGARITHMS OF MID POINTS OF GRADE INTERVALS

FIG 14:

values. The resultant frequency histogram, in the limiting case when the grade intervals are made sufficiently small, merges into a smooth curve called a frequency distribution curve. A typical ore distribution frequency curve is shown in Fig. 13. The most distinctive features of the curve are the extreme skewness towards the left, and the asymptotic approach to the X-axis in the range of the higher grade intervals.

This curve is called the <u>lognormal</u> curve, and as the name implies, it is related to the well known symmetrical normal curve of error. By plotting the grade intervals on a logarithmic scale, the symmetrical curve is obtained, as shown on Fig. 14.

Watermeyer (1919) is attributed as being first to realise the distinctive distribution of ore values. He wrote after an investigation of ten gold mines:

> "The skew frequency curve is absolutely the most solid fact resulting from my investigations. It is not dependant upon constant or any other form of errors. It is simply the result of smoothing out the irregular polygons obtained from experience."

Sichel (1947) recognised the skew frequency as approximating to the lognormal curve as mathematically described by Gaddum (1945). Since then the lognormal curve has been used to describe:

1. The gold distribution in most economic reef horizons of the Witwatersrand and Orange Free State gold fields. (Krige, 1951 and 1952).
2. The distribution of uranium in the Ambrosia Lake, N.M. area. (Patterson, 1958).
3. Certain blocks of manganese ore, in Maggie Canyon, Arizona. (Hazen, 1958).

Some other ore distributions which approximate to the lognormal distribution are:

1. Low grade zinc lodes at Broken Hill, N.S.W. (personal communication).
2. Gold values at Kalgoorlie. (King, 1950).
3. Scheelite ore at King Island Scheelite Mine, Tasmania. (Ludbrook, 1950).

At the risk of making an over-generalisation it appears that the lognormal distribution is the most satisfactory curve to fit any distribution of ore values in a deposit. The assumption that any particular distribution is lognormal can be tested by a simple statistical test, known as the Chi Square test.

The true average grade of any ore deposit, is the arithmetic mean of the value of all the parcels of ore comprising the deposit. From a few samples an estimate must be made of the average grade. The arithmetic mean of the sample values taken is the best estimator of the average grade of a deposit whose values are normally distributed. But for a lognormal distribution the arithmetic mean of the assays is generally not sufficiently accurate as an estimator of the average grade of the deposit, because undue emphasis or weight is placed on the high grade categories. Sichel (1952) proposed the use of a statistical "t" estimator for lognormal distributions. He showed that it gave a more reliable result than the arithmetic mean for the same number of samples from a lognormal distribution. Using this "t" estimator, Krige (1952) showed that it is possible to indicate the confidence which can be placed in the estimated average of a lognormal population.

Any lognormal distribution is determined completely by its mean value and the relative variation between values of the samples.

$$\bar{x} = \frac{1}{n} \quad x$$

$$V = \frac{1}{n} \quad (x - \bar{x})^2 \quad ---- (2)$$

where x and $\bar{x}$ are the natural logarithms of the sample assay and mean sample assay respectively.

n is the number of samples.

V is the variance of the logarithms of the assay values.

V is the main factor governing the confidence limits, which are increased when V is decreased by either increasing "n" or decreasing the deviation from the mean: $(x - \bar{x})$ in equation (2). The normal procedure is to compute $\bar{x}$ and V as each successive exploratory hole has been completed. If a stage is reached in the close pattern drilling where the factor $\frac{(x - \bar{x})}{n}$ is no longer reduced by the addition of more holes, then the average grade has been estimated within practical limits of accuracy for the methods being used. However, additional drilling and sampling will provide more development and mining data, and will also increase the confidence limits, solely on the basis of the increase in the total number of sample analyses ("n" in equation (2)).

A study of the cumulative variance as each hole is drilled, therefore indicates when sufficient sampling has been done, for a given confidence limit.

If it is found that the ore distribution is of a normal type, with only moderate skewness, then the normal statistical formulae can be applied to determine the arithmetic mean, the standard deviation and confidence limits. However it is proposed to assume the more likely and more complex case of the distribution of the uranium values from the Buller being lognormal.

In order to present the technique of statistical analysis it is proposed to work through an example using assumed assay values which approximate a lognormal distribution. One

TABLE 9.

Arrangement of the Set of Hypothetical Drill Hole Sample Analyses.

% U_3O_8 Grade Intervals	·000 to ·049	·050 to ·099	·100 to ·149	·150 to ·199	·200 to ·249	·250 to ·299	·300 to ·349	·350 to ·399	·400 to ·449	·450 to ·499	·500 to ·549	·550 & up
Mid. Point of Grade Int.	·025	· 075	·125	·175	·225	·275	·325	·375	·425	·475	·525	·575
Number of one foot samples	16	34	22	14	11	7	7	5	2	1	1	2
Thousands of tons	333·3	708·3	458·3	291·7	229·2	145·8	145·8	104·2	41·6	20·8	20·8	41·6
Cumulative Tonnage x 1000	2541·4	2208·1	1499·8	1041·5	749·8	520·6	374·8	229·0	124·8	83·2	62·4	41·6

NOTE: Consider the samples as being taken from holes drilled at 500 feet centres.

horizon only is considered since the computations for each horizon or orebody must be done individually and an overall average grade obtained from the means of each horizon.

The first step in the computation of ore reserves from the drill hole assays is to group each sample in the appropriate grade interval. The assumed assay values, of the standard size, one foot drill core samples are shown grouped in their respective grade categories in Table 9. Now if the orebody is thin, the grade of ore mined is not the actual grade of ore in the ore horizon, due to dilution caused by the stoping width being greater than the width of the ore horizon. The minimum stoping width is four feet. Therefore if any ore intersections are less than four feet in width the average assay must be diluted to the equivalent ore grade over the minimum stoping width, before it is entered in the table. The frequency curve and histogram of the 122 one foot sample analyses is shown in Fig. 13.

The statistical arithmetic mean of the values in Table 9 is 0·162% U_3O_8, while Sichel's "t" estimator gives ·166% U_3O_8 as an estimate of the average grade of the orebody represented by the samples. In this case the use of the "t" estimator may seem an unnecessary refinement, but this is not always so.

The standard error of a skewed distribution such as the lognormal, has no practical value. However, confidence limits can be obtained for a lognormal distribution by applying the theory of the notmal distribution to the logarithms of the grade values and then making the transformation

$$\text{from:} \quad \log_e z = x$$

where "z" is the actual grade value.
Using the method described by Sichel (1952), for a 95% confidence:

Upper limit = 0·196% U_3O_8
Lower limit = 0·140% U_3O_8

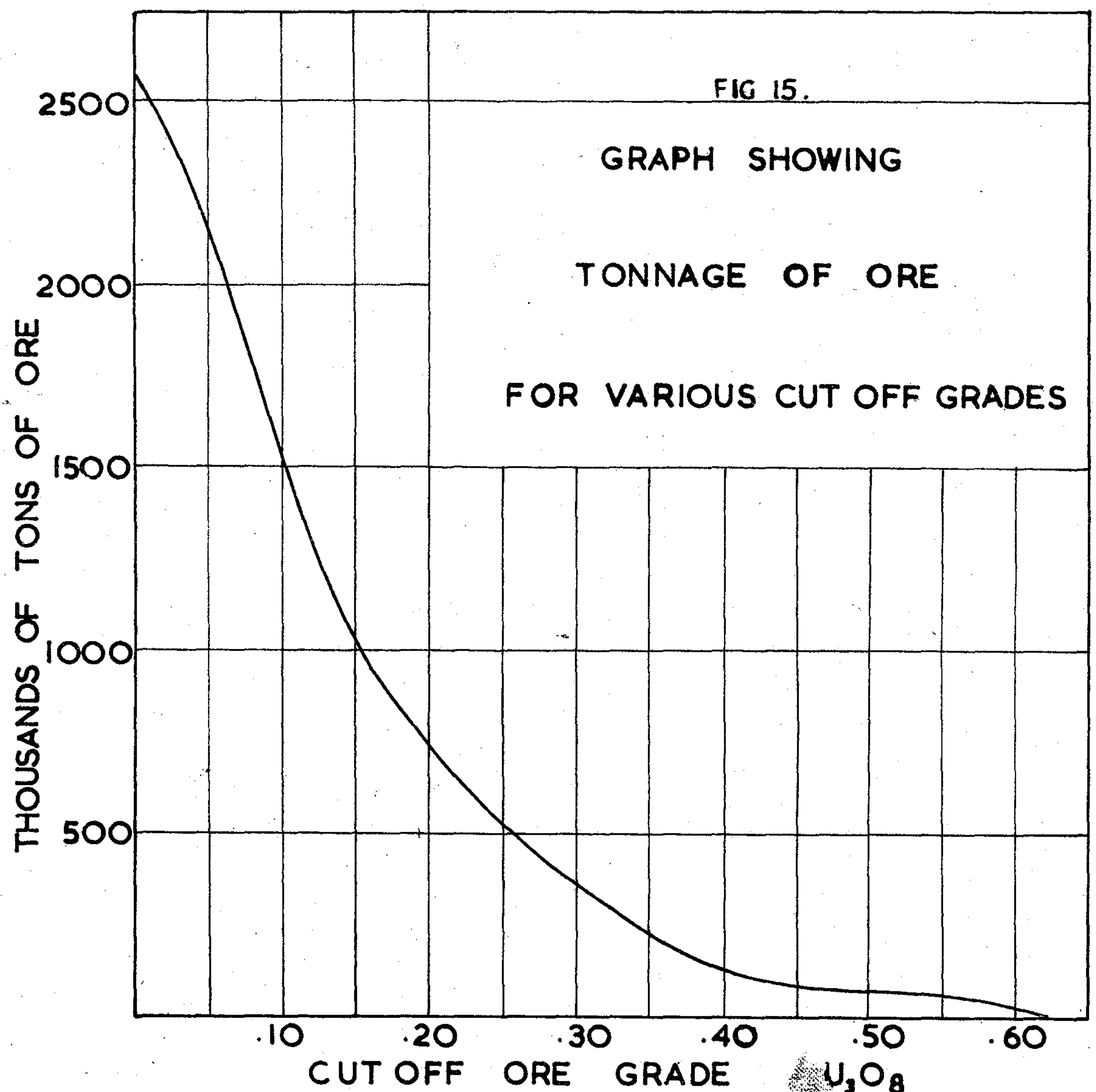


FIG 15.
GRAPH SHOWING
TONNAGE OF ORE
FOR VARIOUS CUT OFF GRADES
THOUSANDS OF TONS OF ORE
2500
2000
1500
1000
500
·10
·20
·30
·40
·50
·60
CUT OFF ORE GRADE U_3O_8

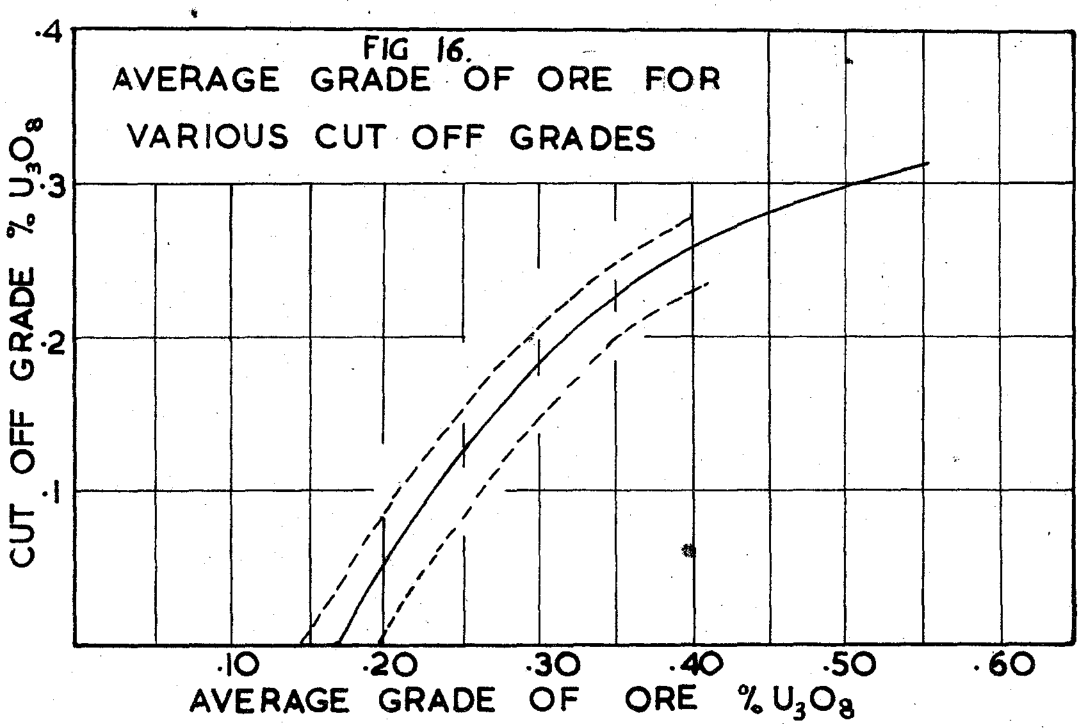


FIG 16.
AVERAGE GRADE OF ORE FOR
VARIOUS CUT OFF GRADES
CUT OFF GRADE % U_3O_8
·4
·3
·2
·1
·10
·20
·30
·40
·50
·60
AVERAGE GRADE OF ORE % U_3O_8

Therefore, on the basis of 122 one foot samples, it is possible to state that the average grade of the orebody has a 95% probability (19 chances in 20) of lying between 0·140% and 0·196%, with a most likely value of 0·166% U_3O_8.

The statistical formulae and calculations required for the results given above may be referred to in the Appendix.

In computing the tonnage from drill hole results a simplification can be made when each sample has an equal "area of influence". This condition has been satisfied by drilling on a uniform grid. With equal areas of influence, tonnage is proportional to thickness: each one foot sample represents an equal tonnage of ore. Therefore in the example taken, the frequency of each grade interval, is directly proportional to the tonnage in the respective grade intervals.

Density of Hawks Crag Breccia	= 12 cu.ft./ton
Area of influence of each drill hole	= 500 x 500 sq. ft.
Tonnage Factor (tonnage represented by each foot of sample length)	$= \frac{250,000}{12}$
	= 20,833 tons/foot

The tonnage and cumulative tonnage of each grade interval is shown in Table 9.

Fig. 15 shows the cumulative tonnage plotted against the cut-off grade of ore, which is the grade such that the direct mining and milling costs equal the value of the ore. The graph indicates the tonnage of ore above any particular cut-off grade. For example, if the cut-off grade of ore is estimated to be 0·15% U_3O_8 the graph shows that there is about one million tons of ore above this grade available in mining.

By calculating the statistical arithmetic mean of the sample analyses above a particular grade, the average grade of mineable ore for a particular cut-off grade is obtained.

Fig. 16 shows this graphically. The 5% confidence limits of the average grade of the deposit are drawn parallel to the average grade line. The graph indicates the average grade and 5% confidence limits that result from any particular cut-off grade. For a cut-off grade of 0·15% U_3O_8, the estimated average grade of ore available in mining is 0·27% with a 95% probability of being between 0·23% and 0·31%.

The technique of plotting tonnage and grade of ore against cut-off grade is used by the U.S.A.E.C., and as Patterson (1958) points out, it is especially useful for analysing low grade, sedimentary type uranium deposits.

5. Diamond Drilling.

Diamond core drilling is superior to other forms of non-core drilling mainly because an actual section of the rock may be observed at the surface. This is a valuable aid to the geologists in interpreting the underground structural environment in which the radioactive horizons occur.

The primary object of diamond drilling is to drill holes as quickly and economically as possible and to obtain the maximum percentage core recovery, of a diameter consistent with accurate sampling of the uranium horizons. Maximum core recovery is not consistent with maximum footage per machine shift and a large diameter core improves core recovery but reduces drilling speed and increases costs per foot drilled.

The optimum size of exploration holes for depths up to 1,000 feet is AX. The approximate diameter of AX size holes is $1\frac{7}{8}$ inches, producing 1 3/16 inch diameter core, which is of sufficient size to give a good core recovery in most formations. It may be found in the Buller that AX size holes do not give a good core recovery when drilling through faults or bands of fine siltstone. If this is the case the hole size should be increased.

The core should be laid out in boxes with the appropriate footages carefully marked. All the geological information found from an analysis of the core should be recorded each shift by a competent geologist. By scanning the core with a ratemeter, mineralized core sections can be easily indicated. The radioactive sections of the core should be split in half longitudinally, one half being retained as a geological record, and the other half, appropriately labelled, sent away for assaying. Care should be taken to ensure that no fines are lost when splitting the core.

TABLE 10.

Comparison of AX Size Diamond Drilling Costs.

Location.	Year.	Average Depth. of hole	Total Drilling Cost per foot	
Copper Belt, N. Rhod.	1950	1,000 feet		62·4/-
Beaverlodge, Canada	1954	175	$4·18	30·4/-
Blind River, Canada	1954	3,500	$7·13	52/-
Blind River, Canada	1954	550	$3·96	28·8/-
Colorado Plateau, U.S.A.	1952	500 - 750	$5·50 to $7·50	40/- to 51/-
Buller Gorge, N.Z. (estimated)		800		84/-

A comparison of diamond drilling costs in Canada, U.S.A. and Rhodesia, is shown in Table 10. The estimated cost of 84/- per foot for the Buller Gorge may appear unduly high in comparison but the following factors must be taken into consideration:

1. The thick forest and rugged terrain make transport and flitting costs high.
2. New Zealand wages are high. The labour costs of the drill crew and other workers directly concerned with the drilling and setting up is estimated at £15 per shift (including holiday pay, workmen's compensation, etc.
3. The heterogeneity of the Hawk Crag Breccia. The variation in the rock makes drilling difficult and gives a reduced bit life. The Diamond Drill Handbook gives a bit life of 40 to 70 feet for conditions similar to those of the Buller Gorge.
4. There are no local facilities for resetting diamond bits.
5. There are few skilled core drillers in New Zealand.

An analysis of the estimated drilling costs is given in Table 11 in which the above factors are considered and a comparison of the conditions with those of known cost schedules is made. An average footage of 15 feet per shift (including setting up time) was also used as a basis for estimating the costs given in the table: this is the average footage per shift quoted in the Diamond Drill Handbook.

TABLE 11.

Analysis of Estimated Diamond Drilling Costs.

Item.	Cost per Foot.
Labour	20/-
Transport and Flitting	20/-
Bits and Reamers	15/-
Supplies and Repairs	15/-
Fuel and Oil	2/-
Depreciation	2/-
Sundries (incl. Administration)	10/-
	84/-

With transport and flitting constituting a major component of the drilling cost, mention should be made of the small portable diamond drill. There are several models obtainable, weighing from 300 to 500 lbs. and capable of drilling to 100 feet, recovering $\frac{7}{8}$ inch core. The portable machine has an application in the final stage of proving when the pattern of ore values is being determined. It could be used for shallow close pattern holes near the outcrops.

The use of a helicopter to facilitate flitting between holes and servicing the rigs, has been proposed. This would reduce flitting costs provided it is om hand when required, and can be kept working at full pay load, since the hire cost is in the range of £80 per hour for a pay load of about 1,000 lbs. There will probably be no more than two rigs in operation and therefore it is unlikely that a helicopter could be kept in constant use. Also, the access road upon which a considerable amount of money has already been spent, would not be required if helicopter transport were used.

6. Radioactivity Logging of Drill Holes.

Gamma ray logging constitutes an ideal method of detecting radioactivity in drill holes. It is also used in determining the thickness and grade of radioactive horizons. In most cases the results of drill hole probing are used in conjunction with drill core or cuttings assays, but in some circumstances the results of drill hole probing alone have been used in the estimation of ore reserves. The advantages of gamma ray logging over core and/or cuttings analyses are the reduced cost of both assaying and drilling, and the fact that the gamma ray log provides a complete profile of the hole.

Gamma ray logging has an important application in the control of ore grade in stoping and underground development. Exploratory holes can be drilled with the normal percussion blasthole machines, the hole probed, and the required information obtained quickly -- all without any reorganisation of underground work.

If, at any stage in the proving programme, it is thought that gamma ray logging and sludge analyses will provide sufficient information for assessing ore grades with no loss of accuracy, then core drilling may be dispensed with. In other words, if the additional cost of drilling is not balanced by the geological information obtained from a study of the core, then waggon drilling which is considerably cheaper, should be undertaken. Gamma ray logging is an alternative to core drilling in ground where core recovery is low.

There are several types of drill hole probe in use. The main components of the logging equipment are a power source, a reel unit to raise and lower the probe, a waterproof probe containing the Geiger tube, an amplifier and ratemeter which amplifies and indicates the number of electrical impulses received through the cable from the tube, and a strip chart recorder that graphically plots the number of impulses received by the ratemeter, against the probe depth.

The rate of movement of the probe is dependent on the degree of accuracy required on the profile chart. Holes are usually logged as the probe is withdrawn, but by logging when both raising and lowering the probe, the position of a radioactive horizon is determined more accurately. Simpson (1950) pointed out that it is possible to accurately determine only the upper limit of the ore when lowering the probe, and the lower limit when raising the probe. This is because of the time lag in the counter circuit.

The logging equipment must be frequently calibrated against standard U_3O_8 values in similar geometry to that in which the equipment will be used. One method is to have a "standard hole", samples from which have been chemically assayed. The probe is tested and adjusted for "background" in this hole each day. The U.S. Geol. Survey used four feet diameter culverts in which simulated orebodies of predetermined grade and thickness were constructed. A two inch test hole in which the probe was tested, was made in the centre of the culvert.

Drill hole probing is subject to certain errors caused by extraneous radioactivity, the most important being radon contamination of the holes.

Obviously, the best way to avoid contamination of the hole is to log it immediately after it is drilled - giving the radon no time to break down into its daughter elements. But this is not always practicable. In an investigation into the effects of radon in drill holes Hilpert and Bunker (1957) found that most holes can be decontaminated by blowing them out with compressed air for 30-60 minutes. They also concluded that although holes were effectively decontaminated by being filled with water, the water reduced both the count rate and the indicated thickness of the mineralized horizon, to less than that obtained in air. The products of thickness and grade figures were reduced by about 20%.

Radon contamination of drill holes in the Buller may not be serious because of the general non-permeability of the breccia, and the fact that the mineralized horizons lie below the water table. Fault shattered rock provides passageways for the radon to migrate away from the mineralized zones and drill holes through faulted areas will be susceptible to radon contamination.

7. Summary: Estimated Proving Costs.

(a) Outcrop Sampling.

Allowing two men an average time of three days (including wet weather) to obtain one reduced bulk sample; labour cost (including wages, holiday pay, field allowances, etc.) at £4 per man shift:

Cost per bulk sample	= £24	
Cost of 14 " "	= £336	
Time taken	= 2 months	£340

(b) Scout Pattern Core Drilling

Total footage	= 12,800 feet	
Cost per foot	= £4: 4: 0	
Total drilling cost	=	£53760

Considering 15 feet as the average footage per machine shift, time taken for scout drilling $= \frac{12,800}{15}$

= 853 drill shifts

Working two shifts per day with one rig, the approximate time taken to complete the scout drilling pattern is 18 months.

(c) Assaying and Geology.

Radiometric equilibrium assay equipment	£1500
One geologist for two years at £2000 per annum (salary plus expenses)	£4000
One assayer or assistant geologist for two years at £1000 per annum (salary plus expenses)	£2000

(d) Surveying.

Location of holes, etc. Part time. Two years at £500 per annum.	£1000

(e) Administration.

Two years at £1000 per annum.	£2000
	£64,600

The above cost summary shows that the initial proving work, of taking bulk samples from the "A" horizon, and drilling the area most favourable for mineralization at 1,000 feet centres, is estimated to cost £65,000.

It is impossible to estimate the total cost of proving the deposit since the design of the third stage of proving (close pattern drilling and/or prospecting adits) is entirely dependent on the results obtained from the scout drilling grid.

Waggon drilling and gamma ray logging would reduce drilling costs appreciably in the final stage of proving.

X. MINING.

1. Introduction.

At the present stage in the development of the Buller uranium field there is insufficient knowledge of the distribution of mineralization, and of the size, grade and attitude of the orebodies to propose a feasible method of mining. The discussion on mining is therefore limited to a general assessment of factors likely to be involved (in the light of present information) if the field is exploited.

Literature on the mining practices adopted in the Colorado Plateau and Ambrosia Lake uranium fields was studied since the coffinite deposits in these areas are, in some respects, a similar mining proposition to those at the Buller. Multi-seam coalmining in the United States, Europe, and Great Britain was also used as a precedent in discussing problems likely to be encountered in the mining of the Buller ore horizons.

A basic factor underlying all discussion on mining, is that the immediate future of the uranium market will almost certainly result in depressed prices. Consequently, elaborate high cost methods of working (such as resuing, stowage support etc.) were not considered.

2. Size of Mine Output.

The first consideration in designing the mining practice is the estimated output per day. The tonnage of proved and probable ore must be sufficient to amortize the capital expenditure on the mine and mill, with a reasonable interest on the capital invested.

In general the largest output that can be obtained results in the maximum profit. For as Parks (1949) states:

> "the shorter the period for mining a given reserve the greater will be the specialisation and therefore the greater the economy in production, the lower will be the fixed charges per unit and the sooner the profit to be derived will be realised."

The costs incurred in any venture may be grouped in three categories:

1. Direct Costs.

   These costs are directly proportional to the tonnage of ore produced; they include miners' wages, expenditure on explosives, timber, drill bits etc.

2. Fixed Costs.

   These costs are completely independent of the rate of production. There are few costs which can be grouped under this category in mining. The cost of proving the orebody is in this category.

3. Indirect or Semi-variable Costs.

   These costs vary with the size of the mine output but are not proportional to the rate of production. They usually remain constant over a limited range of production and increase at a lesser rate than direct costs, with increasing production. It is because these costs are not directly proportional to the rate of production that a larger operation shows a greater profit per ton. Ore transport, ventilation, administration, plant and equipment are in this category.

Theoretically the most economic rate of production is infinitely large, but with large organisations efficiency drops, the labour problem is more difficult, administration unwieldy, and the capital requirements become very large. In addition the following two major factors must be assessed before deciding the scale of operations:

1. Output of the Mill.

The capital cost of any uranium mill is high and the capital cost per ton milled is significant. A study of the Canadian uranium mills shows that the capital cost of the mills, per ton of capacity does not vary significantly between milling capacities of 500 tons per day and 5,700 tons per day. From this it would appear that the optimum mill capacity is not critical. The calcite gangue, the fine particle size, and the low grade ore of the Buller may necessitate a large output, to reduce the "indirect" milling costs, and thus enable a margin of profit to be maintained.

2. Market Considerations.

Since the market is limited, the mine production may be decided by the amount of U_3O_8 which can be profitably sold.

Because of the threatened capture of the nuclear power market by thorium breeder reactors and the fusion process, it appears that the demand for uranium may fall in the 1980's. From this point of view it is desirable that the deposit is exhausted in ten to fifteen years at the most, depending upon the date of commencement of production.

Considering depreciation of assets, such as buildings, equipment and plant, and for stability of employment, a mine life of at least ten years is desirable.

Taking ten years as the estimated life of the mine, and knowing the ore reserves, a maximum rate of production can be estimated, and this, as stated earlier, is the most economic rate of output. The mine and mill can then be designed for this tonnage per shift.

3. The Effect of the Distribution of Mineralization on Mining.

Although the uranium occurs in a sedimentary environment, the surface geology indicates great lateral variations in the thickness and grade of the ore. When the ore has been proved, it may be found that the oregrade mineralization occurs sporadically throughout the Tiroroa Facies and the ore-bodies are separated from each other by comparatively large distances, both laterally and vertically. In this case, the value of each orebody may be insufficient to cover the cost of developing and mining it, from one set of underground workings. A number of small mines may be necessary to exploit the uranium field.

The advantages of a number of small mines over a single large undertaking for a number of isolated orebodies are:

1. Small isolated pockets of ore are mined economically.
2. Labour problems are reduced. The mine workers could be the lessee in the form of a syndicate.
3. The mines are developed on a small capital outlay.
4. The ore is produced for a small working capital. That is, the ratio $\frac{\text{Indirect Cost}}{\text{Variable Cost}}$ is relatively low.
5. Supervision of the workings is easier.
6. The system is flexible, therefore any changes in mill requirements can be met quickly.
7. The amount of dilution of ore is decreased with small operations.

This system of a number of small mines is worked with considerable success on the Colorado Plateau, in areas where small uranium deposits occur. For instance, the Four Corners Uranium Corporation (see ref. 64) operate eight mines on this system. The company drills and delineates an orebody, then lease the mineral property. The lesses operates under a "split-cheque" system, the Company receiving between 40% and 60% of the ore receipts.

By operating a number of small mines, feeding a custom mill and treatment plant, the capital requirements are reduced and labour is easier to handle -- both important advantages in any New Zealand industry. However, it will only be economic if the orebodies are isolated, since a large mine is generally more efficient than a small one. Under these circumstances a major company would be responsible for the exploration, transport and treatment of the uranium ore. Isolated orebodies could be leased out to syndicates or small companies for mining; the lessee being paid on the U_3O_8 content of the ore. The parent company could provide the technical mining and geological staff to ensure that wasteful, unsafe mining practices are avoided.

4. Open-cut Mining.

Due to the surface topography, the Dee Point ridge - Trig C area and the dip slope of the Radioactive Creek area are possible sites for open-cut mining, if ore is proved close to the surface.

The economics of open-cut mining are governed by the economic ratio or ratio. This is the number of cubic yards of overburden that can be removed in order that one ton of the exposed uranium ore may be sold at a profit (McGregor, 1953). Open-cut mining is advantageous when the total extraction cost per ton of ore equals or is less than the total cost of underground mining. But even if the unit extraction cost is higher than that for an underground method, open-cut mining may result in a higher total profit if a more complete extraction of the ore is made.

The following factors favour underground mining rather than open-cut at the Buller:

1. Stripping costs would be high because of the hardrock overburden.
2. Any exposed uranium ore is susceptible to leaching because of the high rainfall. Consequently stripping would have to be done simultaneously with the removal of the ore. Due to the high rainfall working conditions would be unpleasant. Also as McGregor (1953) points out, there is sufficient rain on the West Coast to hamper most mobile stripping equipment in winter.
3. In the Radioactive Creek area the rocks dip at 35° which is too steep a grade for power shovels, trucks, scrapers, etc.
4. There appears to be no distinct parting at the hanging wall and foot wall of the ore horizons. With the large scale methods of rock breaking used in open-cut mines, some dilution of the ore would be unavoidable.
5. The heavy forest cover would increase stripping and transport costs.

It appears from the above that open-cut mining has only a doubtful application in the Buller; the only possibility being if sufficient ore is found near the surface in the Trig C area.

5. Mining Orebodies Occurring at Different Elevations.

The distribution of the uraniferous mineralization discovered so far, indicates that the mining methods employed must be applicable to horizons occurring at various elevations.

If the underlying horizon is mined first, the two most consistent places (according to Stemple, 1956) for serious disturbance in a "superjacent" orebody are over isolated pillars or small groups of pillars left in place in the underlying orebody, and in areas above the line between completely mined areas and solid ore in the underlying horizon.

If the overlying orebody is mined first, the cohesion of the overlying rock mass is destroyed, and excessive weight stresses are transmitted to the underlying horizon. The stress distortions act vertically downwards, and, therefore, fracturing occurs beneath pillar lines, isolated pillars, etc.

There is also the problem of support which Beringer (1928) indicates to be as follows:

> "If the upper one of two parallel lodes is stoped first, can the artificial supports in the lower stope hold the weight of the ground lying between the two deposits? If the lower is stoped first can the roof be so supported that the footwall of the stope not be loosened?"

The National Coal Board after an investigation of multi-seam coal mining (see ref. No. 61)built up the theory of the <u>pressure arch</u>. In any stope a pressure arch is developed with abutments resting on the solid rock on either side. If the width of the stope is increased, the pressure arch increases to a maximum when the roof load can no longer be transferred to abutments around the solid rock perimeter. This is called the width of the maximum pressure arch and for widths greater than this, one abutment is on the stope face and the other in the caved area. The pressure arch moves with the stope face, and therefore the distance from the face to where waste is consolidated is maintained constant.

The destressed zone around the stope is ellipsoidal in shape, with the height about twice the width (according to N.C.B.).

The N.C.B. estimate the width of the maximum according to the formula: $W = \frac{D}{20} + 20$

where W = width of the maximum pressure arch in yards

D = depth in feet

They give the width of the concentrated abutment loads as 20 to 30 yds.

Using the pressure arch theory it is possible to assess the effects in a given horizon on workings above and below. The probable zones of concentrated abutment pressure can be indicated on plans, and the stopes planned to avoid these zones.

For mining the comparatively flat dipping Buller horizons there appears to be two alternative methods of mining.

A system of partial recovery may be utilized whereby sufficient pillars are left to provide adequate support for the adjacent horizon. This is the method of mining (room and pillar) used on the Colorado Plateau. Where possible, pillars of "lean" ore are left as support. But it is essential in this method that the pillars are systematically spaced and are of sufficient size to avoid stress concentrations. If the distance between horizons is relatively small, columnization of workings is advisable, i.e. pillars formed vertically over pillars in the lower orebody. Columnization of pillars is considered to be particularly important if mining is done simultaneously in several horizons.

In partial extraction the hanging wall rock fails due to weakness, and the subsidence consists of a fall of this failed rock along the opening, to a height ranging from a few inches to several tens of feet. This subsidence, therefore, would have little effect upon an overlying ore horizon unless it lay within the scope of the strata reached by the fall.

If total mining is attempted complete extraction must be carried out; pillars must be blasted out even if they are not of ore grade. A period of time is necessary for the void created by complete mining to become filled and conditions become stable enough in the adjacent rock. Stemple (1956) reports periods of 5 to 10 years in the Pennsylvania coalfields, for coal seams 5 to 6 feet thick. Total extraction is especially important for thin orebodies, where a given output can be obtained for a smaller advance than in pillar mining. The effect of caving also decreases with decreasing thickness of the orebody.

Longwall mining results in complete extraction, but it is a rigid cyclic system of mining, unsuited to variable distribution of ore values. Retreating longwall mining causes a gradual bending of the overlying strata, allowing it to settle slowly. However the Hawks Crag Breccia is not well bedded, and it probably will not bend, but shear, and cause breaks in the roof rock.

Total extraction is possible in room and pillar mining. The pillars must be extracted in line to relieve them of as much weight as possible, and the superincumbent rock allowed to break and fall in the extracted area.

Stemple (1956) reports that if done carefully and depending on local conditions encountered underground, the horizons can be mined in either ascending or descending order with an equal chance of success. Simultaneous mining of the horizons is generally more difficult, especially where complete extraction is carried out.

From the foregoing discussion, it appears that longwall working and full caving has disadvantages in mining the Buller orebodies. A form of room and pillar mining, with or without pillar extraction, appears at this stage to be the best method of working. But the final choice of the mining method should not be made until underground conditions, the nature of the orebodies are known, and experimental stopes have been designed and tried out.

6. Other Factors in the Development of the Mine.

Fig. 2 shows the proposed location of the mill and mine surface plant. It is the only site close to the mineralized area, where a considerable amount of excavation is not necessary for construction of the plant, but the topography is sufficient for gravity to be used advantageously in the design of the mill flowsheet. The ground is high enough for the plant to be out of danger when the Buller River is in flood. The site is close to the existing railway line but to facilitate road transport a bridge would have to be constructed across the Buller River.

However if the mill is to treat ore other than from the Buller Gorge, the best site for the mill would be at the mouth of the Buller Gorge. This site is centrally located for ore coming from the Bullock Creek, Fox River areas and is closer to Westport obviating the need for housing construction.

The initial consideration in the mine development is to reach the desired ore horizon, provide a ventilation circuit, haulageway and the necessary points of attack for stoping operations to begin. A possible mine entry is shown in Fig. 17 which is a section along the line XY on Fig. 2. The proposed main entry is an adit driven at a rising grade of 1 in 200 to facilitate drainage and haulage, and is the main haulage level of the mine. Where the ore horizons are situated in a hill, above the level of the surrounding country mine entry by level adit is considered to be the most economic. Since the main haulageway is beneath the ore horizons the greatest use is made of gravity. The ore is simply dropped down chutes from the stopes to the haulage level.

A shaft sunk from the surface in the position shown on Fig. 17 completes the ventilation circuit. The difference in elevation between the shaft collar and main haulage portal provides natural ventilation which may have to be supplemented by a fan as the mine workings extend. Development work in providing points of attack to the ore horizons can be done both from the ventilation shaft and the main haulage level.

Locomotive haulage on a 3 feet gauge well ballasted track is generally recognised as the most economic form of main line level haulage. Where the ore body is thin and dips only at a few degrees, scrapers are the most economic form of secondary haulage; they could be used for scraping from the stopes to the ore chutes.

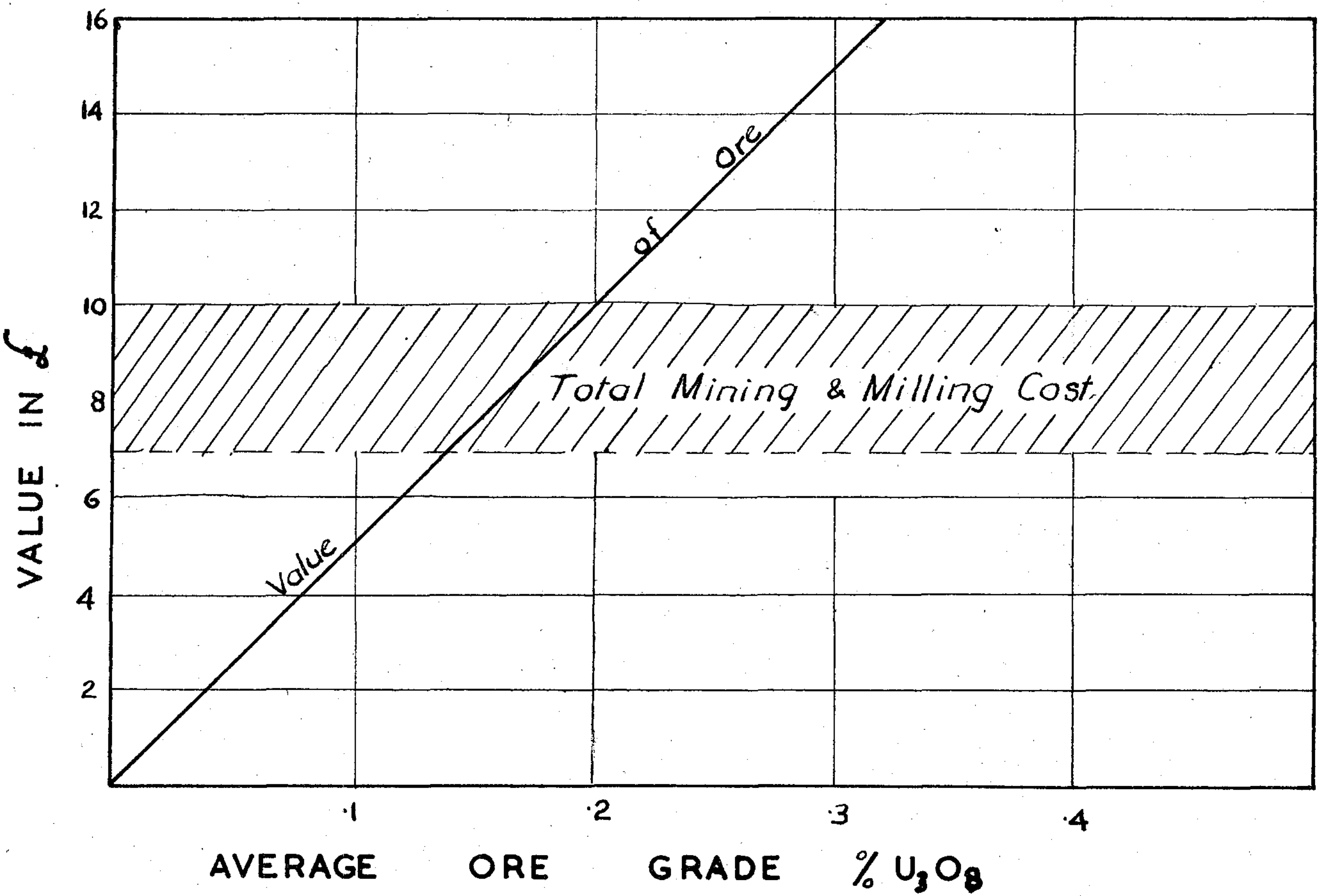


FIG. 18.

N.B. Mill recovery of 90% used in calculating value of ore.

In driving the finger raises, ore chutes, and in pillar extraction use should be made of the longhole method of blasting. It has proved more economical and safe than conventional methods for raises up to 100 feet in length. This method is now worked successfully in many metal and coal mines throughout the world.

Other factors of importance to be incorporated in the design of the mine to reduce the radiation hazard were discussed previously under the "Health Hazard of Uranium Mining.".

7. <u>Estimated Oregrade Required</u>.

From a study of mining and treatment costs in the Colorado Plateau, Ambrosia Lake (Youngberg, 1958), Witwatersrand and Blind River (Skinner, 1958), uranium fields, an attempt has been made to estimate likely working costs in the Buller uranium field. The costs were estimated by comparison, and adjustments made when it was thought that conditions in the Buller differed from those in the known cost schedules. No attempt was made to give a detailed cost analysis, since there is sufficient knowledge of the uranium field at the present time. But the overall figures given probably contain many compensating errors and should provide a guide as to the costs likely to be incurred in exploiting the Buller.

The estimated costs are:

Total Mining Cost	£4:10: 0 to £6:10: 0	per ton of ore
Total Milling Cost	£2:10: 0 to £3:10: 0	" " " "
Total Cost	£7: 0: 0 to £10: 0: 0	" " " "

These figures are represented on Fig. 18 which is intended to serve as an indication of the average grade of ore likely to be required for the uraniferous mineralization to be considered as a mining proposition. A selling price of the equivalent of $7.00 per pound of mill concentrates, formed the basis of the ore value calculation.

XI. SUMMARY AND RECOMMENDATIONS.

It is recommended that proving be undertaken. If the surface samples prove favourable then the scout drilling programme consisting of about two years' work should be undertaken. The estimated cost of this proving work is £65,000. If the existence of a workable deposit is indicated by the scout drilling programme, then efforts should be directed to providing a market for the uranium, before any further proving and development work is attempted.

The uranium market trend is towards depressed prices until about 1972 when the nuclear power industry is expected to provide a major market for uranium. At about that time the uranium price should be stabilised in a free market, and further expenditure in the Buller may be warranted. With the results of the scout drilling known, the Buller will be in a favourable position should any sudden increase in demand occur or an international trade agreement enables the uranium to be marketed.

It is considered that the risks of oversupply of U_3O_8 are too great to justify more than the expenditure of £65,000, as a risk capital investment at the present time. In the period 1962-72, at 5% interest rate £65,000 will amount to £106,000 which sum must be recovered when productive operations commence.

For assaying the uranium samples the radiometric equilibrium method is recommended.

Whether the existing legislation regarding the mining and concentrating of uranium is sufficient or not, will be proved only when a uranium mining industry is established. Conditions are made reasonably attractive, however, by the existing legislation, for a company planning to carry out uranium mining operations in New Zealand.

Income taxation as applied to companies mining uranium in New Zealand is less demanding than in most countries of the world. The taxation laws show a fair understanding of the inherent risks of mining investment.

An incentive to attract capital would be to abolish the royalty payable on uranium.

There is a potential danger to human tissue from mining uranium ore. New Mining Regulations would be necessary to ensure that the maximum tolerance of radiation from the uranium ore is not exceeded. But the provisions for ventilation and dust control in most metal mines are sufficient to eliminate this danger.

By careful planning and stope control, the various uranium horizons can be mined successfully, although 100% extraction will probably not be achieved.

For the Buller uranium deposits to become an economic mining proposition, ore at over 0·2% U_3O_8 average grade, must be proved.

REFERENCES.

1. Bailey, J. E., 1958. Preplanning Safety in the Uranium Mining Industry, AIME Preprint No. 5820A2.

2. Bale, W. F., and Shapiro, J. V., 1956. Radiation to Lungs from Radon and its Daughter Products, Proc. Int. Conf. Peaceful Uses Atomic Energy, Vol. 13, 233-236.

3. Beck, A.C., Reed, J.J., and Willett, R.W., 1958. Uranium Mineralization in the Hawks Crag Breccia of the lower Buller Gorge Region, South Island, New Zealand, N.Z. J. Geol. & Geophys., Vol. 1, No. 3, 432-449.

4. Beckerly, A. J., 1953. Exploration for Nuclear Raw Materials, Macmillan and Co. Ltd., London, 259.

5. Beecham, A., 1956. The Position of Coal in the New Zealand Economy, Proc. Min. & Quarry Conf. Otago Univ., Vol. III, No. 113, 118.

6. Beringer, B., 1928. Underground Practice in Mining. Mining Pub. Ltd., London, 101.

7. Carman, J.S., 1957. The Future of Uranium. Can. Min. Journ., Vol. 78, No. 6, 148-150.

8. Cumming, J. D., 1956. Diamond Drill Handbook, Smit & Sons, Toronto, 2nd Ed., 358-360.

9. Dare, W.L., Lindblom, R.A., & Soulé, J.H., 1955. Uranium Mining on the Colorado Plateau, U.S.B.M., Inf. Circ. 7726.

10. Dickinson, S.B., 1955. Annual Report of Director of Mines, South Australia, 34.

11. Eicholz, G.C., Hilborn, J.W., & McMahon, C., 1953. The Determination of Uranium and Thorium in Ores, Canad. Jour. Phys., Vol. 31, 613-628.

12. Franklin, 1958. Uranium, Eng. & Min. Jour., Vol. 159, No. 2, 122-124.

13. Gaddum, J.H., 1945. Lognormal Distributions, Nature, Vol. 156, 463-466.

14. Goodlet, B.L., 1958. Planning for Nuclear Power, Optima (Quarterly Rev. publ. Anglo-American Corp. of S. Afr. Ltd.), March, 35-41.

15. Greaves, M.C., 1957. A Beta-Gamma Counting Method for Radiometric Determination of Uranium, Proc. Aust. Inst. Min. & Met., No. 182, Pt.1, 23-36.

16. Hagart, R.B., 1957. National Aspects of the Uranium Industry, Jour. Sth. Afr. Inst. Min. & Met., Vol. 57, 565-579.

17. Hazen, S.W., 1958. A Comparative Study of Statistical Analysis, U.S.B.M. Report of Invest. 5375, 29-65.

18. Hilpert, L.S. & Bunker, C.M., 1957. Effect of Radon in Drill Holes on Gamma Ray Logs, Econ. Geol., Vol. 52, No. 4, 438-455.

19. Hodge, H.C., 1956. Mechanism of Uranium Poisoning. Proc. Int. Conf. Peaceful Uses Atomic Energy, Vol. 13, 229-230.

20. Holaday, D.A., 1957. Control of Radon and Daughters in Uranium Mines and Calculations on Biologic Effects, U.S. Dept. Health, Education & Welfare.

21. Holland, H.D., 1958. The Use of Leachable Uranium in Geochemical Prospecting on the Colorado Plateau, Econ. Geol., Vol. 52, No.2, 190-209.

22. Holmes, S.W., 1958. Uranium-Bearing Conglomerates of the Blind River Algoma Area, Can. Min. Jour., Vol. 79, No. 4.

23. Hutchinson, W.S., 1958. Uranium - Annual Survey, World Mining, Vol. II, No. 5, 95-96.

24. Jamnet, H., & Pradel, J., 1956. The Problem of Radon in Uranium Mines, Proc. Int. Conf. Peaceful Uses Atomic Energy, Vol. 13, 219-221.

25. Keys, D.A., 1958. Peaceful Developments of Atomic Energy, Canad. Min. Jour., Vol. 79, No. 4, 101-102.

26. King, H.F., 1950. Distribution of Assay Values in some Western Australian Gold Ores, Proc. Aust. Inst. Min. & Met., No. 156-157, 135-137.

27. Krige, D.G., 1951. A Statistical Approach to Some Basic Mine Valuation Problems on the Witwatersrand, Jour. Chem. Met. & Min. Soc. Sth. Afr., Vol. 52, 119-139.

28. Krige, D.G., 1952. A Statistical Analysis of Some of the Borehole Values in the Orange Free State Goldfield, Jour. Chem. Met. & Min. Soc. Sth. Afr., Vol. 53, 49-50.

29. Landsell, N., 1958. The Atom and the Energy Revolution, Penguin, 113-146.

30. Ludbrook, N.H., 1950. Statistical Analysis of Sampling and Assaying Results, King Island Scheelite Mine, Tasmania, Proc. Aust. Inst. Min. & Met. No. 156-157, 109.

31. Luttrell-West, L.W., 1952. Discussion - A Statistical Analysis of some Borehole Values in the Orange Free State Goldfield, Jour. Chem. Met. & Min. Soc. Sth. Afr., Vol. 53, 67.

32. McGregor, G.W., 1953. Some Factors Affecting Ratio in Coal Mining, Proc. Min. Conf., Univ. of Otago, No. 39, 1.

33. McLean, C.S., 1954. The Uranium Industry of South Africa, Jour. Chem. Met. & Min. Soc. Sth. Afr., Vol. 54, No. 10.

34. Miller, T.F., 1956. Health Protection of Uranium Miners and Millers, Proc. Int. Conf. Peaceful Uses Atomic Energy, Vol. 13, 242-247.

35. Nininger, R. Dr., 1957. Uranium - Annual Review for 1956, Eng. & Min. Jour., Vol. 58, No. 2, 90-92.

36. Paine, L. G., 1957. Uranium - Annual Review for 1956, Min. Jour., May, 51-53.

37. Parks, R.D., 1949. Examination and Valuation of Mineral Property, Addison-Wesley, Cambridge, Mass., 3rd Ed., 145.

38. Patterson, J.A., 1958. Statistical Method of Calculating and Analyzing Ore Reserves, AIME, Pre Print 5820A2.

39. Pennycuick, S.W., 1958. Science for the People, Evening Star Daily Newspaper, Dunedin, N.Z., July 17.

40. Phair, G., & Levine, H., 1953. Differential Leaching of Uranium, Radium and Lead, Econ. Geol., Vol. 48, No. 5, 358-369.

41. Plowden, E., 1957. How Much Uranium Shall We Need, Optima (Quarterly Rev. publ. Anglo-American Corp. Ltd.), December, 169-173.

42. Sichel, H.S., 1952. New Methods in the Statistical Evaluation of Mine Sampling Data, Trans. Inst. Min. & Met., Vol. LXI, 261-288.

43. Sichel, H.S., 1952. Discussion - A Statistical Analysis of Some Borehole Values in the Orange Free State Goldfield. Jour. Chem. Met. & Min. Soc. S. Afr., Vol. 53, 65.

44. Simpson, D.J., 1950. Radioactivity Logging, Trans. Geol. Soc. Sth. Afr., Vol. LIII, 4.

45. Skinner, W.E., 1958. Mining Year Book, Author, London.

46. Squirrell, D.C.J., 1953. Surface Diamond Drilling Problems and Practice in the Copperbelt Area of Northern Rhodesia, Trans. Inst. Min. & Met., Vol. LXII, 552.

47. Stemple, D.T., 1956. A Study of Problems Encountered in Multiple-Seam Coal Mining in the United States. Bull. Virg. Polyt. Inst., Vol. XLIX, No. 5.

48. Thomson, G., 1957. Uranium in the Production of Power, Optima (Quarterly Rev. publ. Anglo-American Corp. Ltd.), Sept., 111-116.

49. Tsivoglou, E. C., 1956. Use of Radio-Isotopes in Studies of Health Hazards in the Uranium. Mining Industry. Proc. Int. Conf. Peaceful Uses Atomic Energy., Vol. 13, 242-247.

50. Watermeyer, G.A., 1919. The Application of the Theory of Probability in the Determination of Ore Reserves - Discussion. Jour. Chem., Met. & Min. Soc. S. Afr., Vol. XIX, 280.

51. Wellman, H.W., 1950. Ohika Beds and the Post-Hokonui Orogeny, N.Z. J. Sci. Tech., Vol. 32, No. 3, 11-18.

52. Williams, G.J., 1957. New Mineral Discoveries in the Nelson Province, Thos. Cawthron Mem. Lect. No. 32, Stiles, Nelson.

53. Wodzicki, A., 1958. Geochemical Prospecting for Uranium in the Buller Gorge, A.O.S.M. thesis (unpubl.), 11.

54. Worley, R., 1954. Present day Review of the Great Undertakings in New Zealand of the Electricity and Gas Services, Auck. Gas Coy., 2.

55. Young, R.G., & Ealy, G.K., 1956. Uranium Occurrences in the Ambrosia Lake Area, N.M., USAEC.

56. Youngberg, E.A., 1958. The Future of Uranium. World Mining, Feb., 40.

57. Yourt, G.R., 1955. Sampling for Radioactivity in Uranium Mines, Can. Min. Jour., Vol. 76, No.9.

58. 1957. N.Z. Mines Statement. N.Z. Govt., Wgtn., 11.

59. 1956. Schedule of Uranium Drilling Costs, Boyle Bros. Drilling Co. Ltd., Canada, unpubl.

60. 1956. A.E.C. Announces New Uranium Procurement Program, Press Release, No. 150.

61. 1954. Report on the Effects of Workings in Adjacent Seams upon new Developments, Trans. Inst. Min. Eng., Vol. 113, Pt. 4, 391-399.

62. 1958. U_3O_8 - Why the A.E.C. Put on the Brakes, Eng. & Min. Jour., Vol. 159, No. 1, 132-143.

63. 1954. U_3O_8 - Formula for Profits, Eng. & Min. Jour., September.

64. 1956. Four Corners Uranium Prospers by Leasing Small Deposits to Miners, World Mining, Vol. 9, No. 3, 40-41.

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A P P E N D I X

STATISTICAL CALCULATIONS.

1. Statistical Arithmetic Mean.

$$M_z = \frac{\Sigma(f.z)}{N}$$

where M_z = arithmetic mean grade of the samples.

f = frequency.

z = mid-point of grade interval.

N = number of samples.

2. The "t" Estimator.

For a lognormal distribution Sichel (1952) showed

$$\log_{10} t = \bar{x}' + \phi(V) \quad \text{------} \quad (1)$$

where t = estimate of the mean grade of the mineral deposit.

$\bar{x}'$ = mean of the ordinary logarithms of mid points of the grade values.

$\phi(V)$ = is a complicated mathematical function of the variance (V) and it can be obtained from Tables set out in Sichel's report.

The calculation of "t" is made by making the following computations:

1. Compute $\bar{x}'$, the mean of the ordinary logarithms of the mid-points of the grade values.

2. Compute m_2', the mean of the squares of the ordinary logarithms.

3. Deduct the square of $\bar{x}'$ from m_2' and multiply by 5·3019 to obtain the variance of the natural logarithms.

$$V = 5{\cdot}3019 \left[m_2' - (\bar{x}')^2\right]$$

4. Knowing V the value of $\phi(V)$ is found by interpolation from Sichel's Tables.

5. Find $\log_{10}t$ by substitution in equation (1).

6. Anti-logarithms of $\log_{10}t$ gives the required "t".

3. Confidence Limits.

According to Sichel in Krige (1952) an approximation (t') can be made for "t" when $N \geqslant 100$.

$$t' = e^{\bar{x}} + \frac{V}{2} = e^{z}$$

where $\bar{x}$ = mean of the natural logarithms
V = variance of the natural logarithms.

To lay off confidence limits it is necessary to know the mean value and standard deviation of the natural logarithms of the lognormal distribution. By applying the normal frequency curve table to the natural logarithms, confidence limits can be obtained.

It can be shown that z is distributed with exact mean

$$e(z) = \mathcal{E} + \frac{N-1}{2N} \sigma^2,$$

and exact standard deviation

$$S.D.(z) = \sigma\sqrt{\frac{1}{N}\left(1 + \frac{N-1}{2N}\sigma^2\right)}.$$

σ^2 and can be estimated by

$$\bar{\sigma}^2 = \frac{N}{N-1} V$$

$$= \bar{x}.$$

For 95% confidence

Lower limit $= e^{z - 1\cdot 96\ S.D.(z)}$

Upper limit $= e^{z + 1\cdot 96\ S.D.(z)}$

4. Calculation of Hypothetical Example (given in the text).

(1) Grade Intervals % U_3O_8	(2) Mid Point of Grade Interval	(3) Fre-quency	(4) (2) x (3)	(5) Common log. of Mid Pts of G. I. x 100	(6) $(5)^2$	(7) (5) x (3)	(8) (6) x (3)
·000 ·049	·025	16	·40	·3979	·1583	6·3664	2·5328
·050 ·099	·075	34	2·55	·8751	·7656	29·7534	26·0304
·100 ·149	·125	22	2·75	1·0969	1·2034	24·1318	26·4748
·150 ·199	·175	14	2·45	1·2430	1·5450	17·4020	21·6300
·200 ·249	·225	11	2·475	1·3522	1·8283	14·8742	20·1133
·250 ·299	·275	7	1·925	1·4393	2·0707	10·0751	14·4949
·300 ·349	·325	7	2·275	1·5119	2·2853	10·5833	16·0006
·350 ·399	·375	5	1·875	1·5740	2·4774	7·8700	12·3870
·400 ·449	·425	2	·850	1·6284	2·6516	3·2568	5·3032
·450 ·499	·475	1	·475	1·6767	2·8114	1·6767	2·8114
·500 ·549	·525	1	·525	1·7202	2·9591	1·7202	2·9591
·550 & up	·575	2	1·115	1·7597	3·0968	3·5194	6·1936
Totals		122	19·70			131·2293	156·9291

Arithmetic Mean $= \frac{19{\cdot}70}{122}$

$= \underline{0{\cdot}162\%\ U_3O_8}$

Calculation of "t"

$$\bar{x}' = \frac{131{\cdot}2293}{122} = 1{\cdot}07565$$

$$m_2' = \frac{156{\cdot}9291}{122} = 1{\cdot}2863$$

$$V = 5{\cdot}3019\ (1{\cdot}2863 - 1{\cdot}07565^2)$$

$$= 0{\cdot}6653$$

By interpolation from Sichel's tables

$$\phi V = {\cdot}14476$$

$$100 \log_{10} t = 1{\cdot}07565 + 0{\cdot}14476$$

$$t = \underline{0{\cdot}166\%\ U_3O_8}$$

$$S.D.(z) = \sigma\sqrt{\frac{1}{N}\left(1 + \frac{N-1}{2N}\sigma^2\right)}$$

$$\sigma^2 = \frac{122}{121}\ 0{\cdot}6653$$

$$= 0{\cdot}6719$$

$$S.D.(z) = \sqrt{{\cdot}6719\left[\frac{1}{122}\left(1 + \frac{121}{244}.\ 0{\cdot}6719\right)\right]}$$

$$= \frac{{\cdot}6719.\ 1{\cdot}3332}{122}$$

$$= 0{\cdot}08569$$

$$z = \bar{x} + \frac{V}{2}$$

$$= 2{\cdot}30258 \times 1{\cdot}07565 + 0{\cdot}3327$$

$$= 2{\cdot}8094$$

For 95% Confidence

Lower Limit $= e^{2{\cdot}8094 - (1{\cdot}96.)\ {\cdot}08569}$

$= e^{2{\cdot}6414}$

$= \underline{0{\cdot}140\%\ U_3O_8}$

Upper Limit $= e^{2{\cdot}8094 + (1{\cdot}96.)\ {\cdot}8569}$

$\doteqdot e^{2{\cdot}9774}$

$= \underline{0{\cdot}196\%\ U_3O_8}$

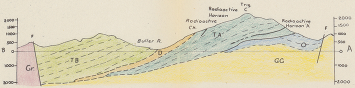


FIG 3 : SECTION ALONG LINE AB ON FIG 2., SHOWING BECK'S INTERPRETATION OF FACIES
Scale : 40 chains to 1 inch.

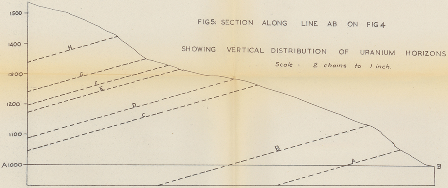


FIG5: SECTION ALONG LINE AB ON FIG4
SHOWING VERTICAL DISTRIBUTION OF URANIUM HORIZONS
Scale : 2 chains to 1 inch.

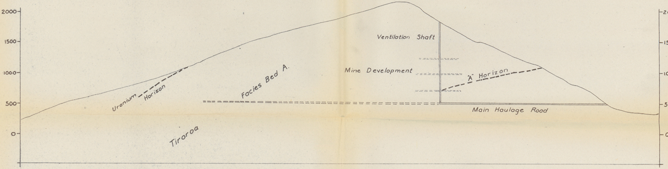


FIG17: SECTION ALONG LINE XY ON FIG 2., SHOWING POSSIBLE MINE DEVELOPMENT
Scale : 10 chains to 1 inch.

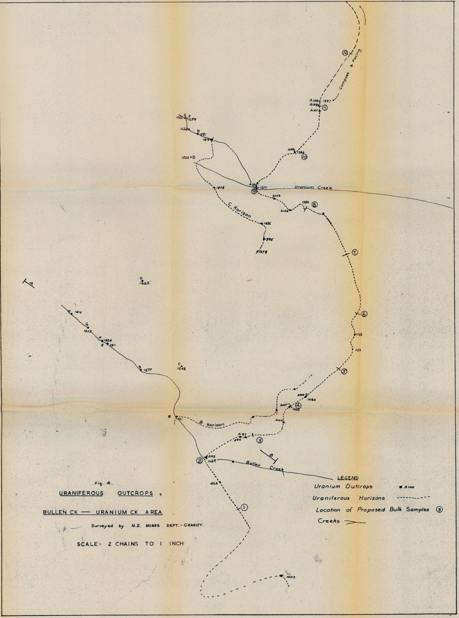


Fig. 4.

URANIFEROUS OUTCROPS,

BULLEN CK — URANIUM CK AREA.

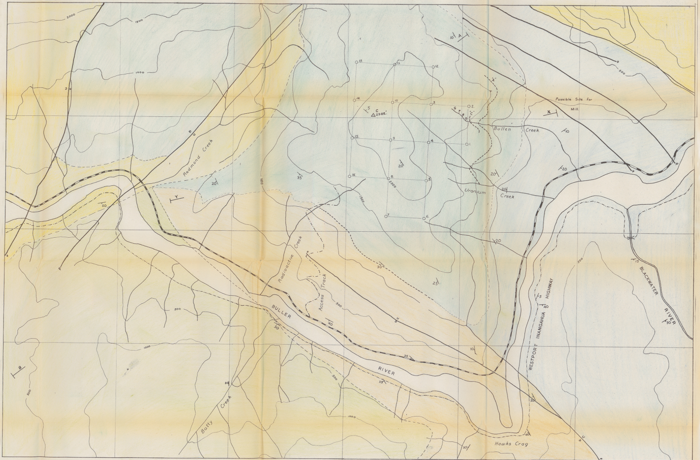
Surveyed by N.Z. MINES DEPT. - GRANITY.

SCALE: 2 CHAINS TO 1 INCH

Fig. 2.

GEOLOGY OF URANIFEROUS AREA
LOWER BULLER GORGE

SCALE: 10 CHAINS TO 1 INCH.



RADIOACTIVE HORIZONS

FAULTS

COLLARS OF PROPOSED SCOUT DRILL HOLES

SECTION LINE

HAWKS CRAG BRECCIA: TIROROA FACIES BED B; DEE POINT "; TIROROA FACIES BED A

OHIKA BEDS

GREENLAND SERIES

GRANITE